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## The New Chemists Building

Elsewhere in this issue the program will be found of the exercises with which the new Chemists Building at 50-54 East Forty-first Street, New York, will be opened on March 17 and 18. For chemistry in America history has been made in the old cozy clubhouse in West Fifty-fifth Street and with it are connected dear reminiscences of many a glorious achievement and many a happy hour enjoyed in good fellowship. Now the old spirit is to be transplanted into the new big and magnificent home. This is the meaning of the opening exercises. The new Chemists Building is a grand monument representative of the united democracy of American chemists. A graceful and thoroughly practical twentieth-century structure, it has been erected, without outside help, to represent the ideals of American chemists. Our hopes and wishes are comprised in the old Saxon miner's greeting: "Glückauf."

## Continuous Copper Converting.

Some years ago we pointed out in these columns the essential difference between a continuous and an intermittent process. This was to the effect that continuous operation was advantageous when the end desired was a large tonnage eliminating impurities cheaply and that an intermittent operation was advantageous when the end desired was a refined or semi-refined product of a definite nature. In short, a continuous process makes for quantity of product and an intermittent process makes for quality of product. For the complete elaboration of this principle the reader may be referred to our Volume III, page 90.

Copper converting is peculiar because it is at once a roughing metallurgical action and a finishing metallurgical action. As it unites two functions in an operative manner it is efficient. But at the same time it has the inherent disadvantages of an attempt to do two things at one time. These peculiar disadvantages are the cost of lining the converter, the amount of power used for furnishing the blast and the danger of loss of values by rapid driving. The lining is before first blow from 20 in. to 24 in. thickness, and, as everyone versed in the metallurgy of copper knows, is made of a plastic silicious material. The lining furnishes the silica for the slag produced by the oxidation of the iron in the matte blown up to blister copper. It is certainly crude to use the lining of the converter as a flux and seems as foolish as nature would if we lived on the inside of our stomach. But like many other processes crude from a theoretical standpoint, it nevertheless, is a practical process, for it works.

Various attempts have been made to use a lining that is basic and hence refractory to the ferruginous slag made. Professor Baggaley made some exhaustive large-scale tests with a basic-lined converter about five years ago at Butte, Mont. Prior to that time Mr. George Westinghouse had conducted a long series of experiments at Pittsburgh and later at Ely, Vt.

For commercial reasons alone both these attempts were never followed out. Indeed, in the several fat years of the metal industry, from 1900 to 1907, improvements were not needed as much as they are now in the present times of diminishing returns.

During the past year large-scale commercial demonstration of a basic-lined copper converter has been made by Messrs. Smith and Pierce at Ely, Nev. Also that enterprising concern, the International Nickel Company, has started work at Copper Cliff, Ontario, on a basic bessemer plant for working up the 30 per cent copper-nickel matte to an 80 per cent matte suitable for smelting by the Orford process. In both these attempts the silica is thrown on the slag continuously, but the tapping of the slag and the blowing are performed intermittently.

It might be argued that the logical process is rather to make the entire smelting process continuous and to do the final refining and the production of the copper anodes in an electric furnace, and it is interesting to give briefly the argument in support of this view. The operation of blowing up a 50 per cent matte to blister copper produces an active scorifying slag that will dissolve to some extent either a magnesite or a chromite brick or tamped basic lining. Oxide of iron, especially when a small percentage of iron sulphide is dissolved with it, is about as corrosive material at high temperatures as is well known. Indeed, oxide of iron can play the rôle of an acid with the formation of ferrates. The proper way, of course, is to use a stationary furnace or converter and water-cool the shell. From a theoretical standpoint this is ideal. Like pyritic smelting and "blast roasting" of sulphide ores the heat balance sheet shows that when once the operation is started at a good rate there is a fair excess of heat generated over heat lost by radiation so that the right conditions can be held uniformly. If this is possible in an intermittent operation there should be no reason why it should not be attained in the right form of a continuous apparatus.

Of course, there is a large proportion of slag—on the average 50 per cent of the original charge—that must be handled. Naturally this is a difficult proposition and regard must be paid to this difficulty. But the great success of the Dwight-Lloyd roasting and sintering apparatus is due to the continuous nature of the process which is conducted in it. It is naturally desirable that continuous copper-converting be done in large units. This would give the operation a sufficient excess of heat so that conditions could not change easily. It is, of course, patent that some means must be used to bring the oxygen of the blast continuously in contact with a large surface of matte. The difficulty and painful slowness of making blister copper out of "white metal" in the old-fashioned reverberatory furnace lay in the fact that the slag protected the matte from the oxygen. Possibly the old Nichols and James reaction of roasted copper-matte on copper-matte with the formation of blister-copper would be applicable. For the final finishing reaction and the removal of the gases nothing can be better or cheaper than the arc-type of electric furnace. The experience in the metallurgy of steel proves this.

We are not indulging at all in prophecy, but only attempting to judge the future from the past. Of late we have seen the great increase in the use of continuous blast-sintering ma-

chines in roasting sulphide ores and the extending use of the reverberatory smelting furnace for fine concentrates and flue-dust. Now we see the basic-lined converter. It is at least not illogical to expect the continuous converter in the not far distant future.

#### Some Interesting Rail Production Statistics.

Statistics of rail production in the United States in 1910, as just presented by the American Iron and Steel Association, show that the year was the next best in the history of the rail trade, the 3,634,029 gross tons produced falling short of the record year, 1906, by 343,858 tons and exceeding by 375 tons the production in 1907, which had stood as the second best year.

Such wild forecasts are sometimes made of the prospects of rail demand, and such curious views seem to prevail in many quarters as to the uses to which our rail production is put, that a cursory review of the general position may be profitable. There was in existence in the United States at the beginning of 1910 in the neighborhood of 400,000 miles of track, laid in steel rails. There was about 350,000 miles of steam railroad track, the remaining 50,000 miles of our estimate covering electric and other public roads using steel rails and industrial tracks. The tonnage of rails in this track can only be approximated, but was probably not far from 50,000,000 tons, which would be sufficient to lay 400,000 miles of track with 80-lb. rails. Up to that time there had been produced in the United States about 60,000,000 tons of steel rails, of which about 2,750,000 tons were by the open-hearth process. In the early years there were heavy imports of steel rails, while in late years there have been heavy exports, and these balance roughly. The deficit of some 10,000,000 tons represents, very roughly, the tonnage which has worn out. The total tonnage produced includes some material which passed through the mill twice, having worn out in service and being rerolled to pass through service again.

It may be of interest to observe, *en passant*, that there was produced in the United States a total of a little more than 10,000,000 tons of iron rails, and a small portion of that tonnage survives, there being at the beginning of 1910 about 8800 miles of steam-railroad track still laid with iron rails, involving, with the iron rails surviving in mines and other industrial plant, well in excess of 500,000 tons.

The fact that the tonnage of steel rails worn out in the United States has been in the neighborhood of only about 10,000,000 tons should assist in dispelling the curious view which is evidently entertained in some quarters that the chief use to which our rail production is put is the replacing of rails which have worn out in regular service. That is not the case. Even of the rails which have become unfit for service a large part has not yielded under proper usage, but has required replacement because too light for the much heavier service now involved. Had it been the case that the rail production of 10, 15 and 20 years ago was required to make up for the wearing out of rails, the present demand would be enormous, for the freight ton-mileage of the country—the only statistical measure we have of the service put upon rails—has approximately doubled each decade.

While there are no precise data as to the mileage of new

track laid in 1910 data are available which support an estimate of approximately 10,000 miles, which if laid in rails averaging 85 lb. per yard would involve about 1,335,000 tons. Exports amounted to 353,180 tons, while of light and standard rails used for industrial purposes the quantity may be estimated at about 250,000 tons. This would leave 1,696,000 tons for all other purposes, the summary being as follows:

	Tons.
10,000 miles steam track.....	1,335,000
Industrial uses .....	250,000
Exports .....	353,180
All other .....	1,696,000
Total production .....	3,634,029

The balance found by difference, 1,696,000 tons, must cover rails used for trolley and other transportation lines outside of steam roads, probably about one-third of the total, the balance being the tonnage used by the steam roads for replacement purposes. The major part of this is now ordinary replacement, but a portion represents the replacement of light sections which cannot stand up under the heavy duty imposed by high speeds and heavy cars and locomotives.

A few of the railroad systems present statistics of the tonnage of rails used for replacement purposes. By analyzing these statistics it may be shown that, very roughly, a ton of rails is worn out for every 250,000 ton-miles of freight traffic. The factor is only an approximation, but has value when there is practically no other indication. The total freight ton-mileage of the country in 1907 was 233,137,000 and in 1909 it was 227,199,000, so that a rough-and-ready rule is that when rails are of section sufficient to escape abuse, and meet only legitimate wear, the steam roads of the country are wearing out about 1,000,000 tons a year.

The study of rail statistics merely for the purpose of ascertaining where the rails go can have only an academic interest. We have made this little study for a more serious purpose. In the past rails have been used chiefly for laying new track; in the future they will be used more and more for replacing other rails which have worn out. The period of greatest railroad building is past; as far back as 1887 occurred the greatest building of new road in a year, and while there remains a great deal of double-tracking to be done the laying of new track cannot be expected to increase much, if at all. The tendency to wear rails out, however, increases steadily. As already observed, there has been, approximately, a doubling of the ton-mileage of freight movement every decade and while this rate of increase may not be continued, there is certain to be increase at some rapid rate.

A rapid increase in freight movement and a slow increase in track mileage obviously involve an increase in the density of traffic on given stretches of track, and we have merely expressed, in terms of the entire railroad system, that which is common knowledge as to individual railroads. As the traffic increases and the rails are subject to greater and greater annual wear, the annual sinking fund necessary to replace the rail increases. It would be a simple matter to write an algebraic formula to illustrate the principle we wish to emphasize, but this form of expression is not necessary. It need merely

be remarked that when the wear on rails was light, and they accordingly lasted a long time, it did not profit railroads to pay a given premium to obtain a more lasting rail, because the premium would increase the annual interest charge, while with a long life even the poorer rail would require but a small sinking fund to provide for its ultimate replacement. As the wear upon rails becomes greater and their prospective life is shortened, the annual sinking fund increases and it becomes more and more desirable to use a better rail, gaining more in decreasing the annual sinking fund for ultimate replacement than is lost by increasing the interest charge.

Thus, while a few years ago the railroads were interested merely in getting a fairly good and a safe rail, now those with the greatest density of traffic are anxious to obtain a very durable rail and are willing to pay a premium in order to secure it. We stand at the threshold of a great development in the manufacture of special steel rails. Only for 1909 and 1910 are there statistics of production of special rails. These statistics are as follows. They are quoted as given, though it is not clear why a distinction should be made between titanium and ferrotitanium steel rails:

1909.	Gross Tons.
Titanium .....	35,945
Nickel-chrome .....	12,287
Nickel-steel and electric.....	1,464
Manganese .....	1,028
Total .....	50,724

1910.	Gross Tons.
Titanium .....	195,408
Electric .....	4,210
Ferrotitanium .....	532
Manganese .....	390
Nickel, chrome and vanadium.....	81
Total .....	200,621

Of the 1909 tonnage of special steel 35,699 tons, or 70 per cent, was made from Bessemer steel, while in 1910 the proportion was 87 per cent. We are not prepared to draw positive conclusions from the statistics. They are suggestive, but perhaps of negative rather than positive conclusions. The high percentages of Bessemer rails treated may mean that open-hearth rails are regarded as "good enough"; at any rate they show that as to the refining action of titanium (the "titanium rail" is not a titanium alloy steel, but a steel purified by titanium, the titanium compounds formed passing into the slag) the Bessemer metal can be treated satisfactorily. The practical disappearance from 1909 to 1910 of nickel-chrome indicates a loss of interest in this alloy. The electric steel rail showed a large percentage increase, and 4210 tons of electric steel rails produced in 1910 furnishes at least a foundation for observation, being enough to lay more than 30 miles of track. The "titanium rail" shows a remarkable growth, from 35,945 tons in 1909 to 196,000 tons in 1910, and it is regrettable that the statistics do not indicate precisely the nature and extent of the treatment to which the tonnage was subjected. It was not until 1907 that as large a tonnage of open-hearth steel rails was produced, yet the open-hearth steel rail was regarded as a factor and was provoking a great deal of discussion.



### Program of the Opening of the New Chemists Building in New York City

The celebration of the opening of the new Chemists' Building, 50 to 54 East Forty-first Street, New York City, will take place on Friday, March 17, and Saturday, March 18.

On Friday, March 17, there will be a reception at the new club building from 3 to 6 p. m.

The opening exercises in the auditorium under the auspices of the Chemists' Building Company will take place from 3:30 to 5 p. m.

In the evening a meeting will be held in the auditorium under the auspices of the New York Section of the American Chemical Society, which will begin at 8:30. After an introductory address by the chairman, Dr. Charles Baskerville, papers will be presented by Prof. R. B. Moore, of Butler College, Indianapolis, on the rare gases of the atmosphere and by Dr. Jacques Loeb, of the Rockefeller Institute for Medical Research, on the characteristics of living matter from the physico-chemical point of view.

On Saturday, March 18, a session will be held in the auditorium from 10:30 to 12:30 under the auspices of the New York Section of the American Electrochemical Society. After an opening address by the chairman, Mr. E. A. Sperry, papers will be presented by Dr. W. R. Whitney on mental catalysis and by Dr. E. F. Smith on recent developments of electroanalysis.

In the afternoon a session will be held from 2 to 4 p. m. under the auspices of the New York Section of the Society of Chemical Industry.

After an introductory address by the chairman, Dr. Charles F. McKenna, papers will be presented by Dr. Charles F. Chandler on the history of chemical industry in New York City and by Dr. W. P. Mason, of Troy, N. Y., on the contributions of chemistry to sanitation.

In the evening a subscription banquet will be held at the clubhouse under the auspices of the Chemists' Club.

On Sunday, March 19, there will be in the afternoon, beginning at 4 p. m., a concert by the Olive Meade Quartet.

Owing to the limited accommodations admission to all functions will be strictly by cards and tickets. Ladies are invited to attend all the functions with the exception of the banquet.

### American Electrochemical Society

The nineteenth general meeting of the American Electrochemical Society will be held in New York City on April 6, 7 and 8 (Thursday to Saturday).

A special feature of this meeting will be the fact that it will be the first convention of a national chemical society to be held in the new beautiful Chemists' Building in New York.

The full program of papers, excursions and social functions will be printed in our next issue.

### International Congress of Applied Chemistry of 1912.

The organization of the Eighth International Congress of Applied Chemistry, to be held next year in this country, has now been completed and will shortly be made public. The dates of the congress have been fixed as follows:

The opening meeting will take place at Washington, D. C., on Sept. 4, 1912 (Wednesday). The other meetings, both business and scientific, will be held in New York City, beginning Friday, Sept. 6, 1912, and ending Friday, Sept. 13, 1912.

The President of the United States will be the patron of the congress. Dr. Wm. H. Nichols is the president, Dr. E. W. Morley the honorary president, and Dr. B. C. Hesse, 25 Broad Street, New York City, is the secretary of the congress.

### American Institute of Chemical Engineers

The third semi-annual meeting of the American Institute of Chemical Engineers will be held at Chicago, Ill., from June 21 to 24 (Wednesday to Saturday).

Arrangements will be made to visit a number of the large technical plants in the vicinity. The committees on chemical engineering education and on standardization of boiler tests will have important reports to present.

### The Western Metallurgical Field.

#### Leadville's Celebration.

Probably no more unique celebration was ever held than that which marked the so-called rejuvenation of Leadville by the discovery of zinc carbonate and silicate. The event was combined with the annual celebration of Robert Burns' birthday, although there is no direct connection between the Scotch poet and smithsonite or calamine other than that the Leadville miners are interested in both.

The novel feature of the celebration consisted in a noon-day dinner served to about 300 guests in a stope of the Wolf-tone mine, 850 ft. underground. The Western Mining Company, Mr. S. D. Nicholson, president, was the host of the occasion. The guests were taken down the shaft and through lighted drifts to the banquet-room. The latter was a stope from which about 3000 tons of 33 per cent zinc carbonate had been extracted and shipped to smelters. It was brilliantly lighted and decorated with flags and national colors, and the whole effect was such as to dispel the idea of being underground. In the evening a more elaborate and pretentious banquet was served at the Elks' Club, at which the significance of the recent discoveries were considered.

The largest problem which lies ahead of Leadville, and one of the most importance in connection with the zinc discoveries, is the future treatment of low-grade ores carrying from 10 per cent to 20 per cent zinc. There is a vast quantity of such ore which, at present, can not be handled. The lowest grade material yet shipped at a profit is 20 per cent zinc, and there awaits professional fame and ample reward for the metallurgist who shall devise successful means of treating the lower-grade ores. It is a significant fact that coincident with the discoveries of Leadville's new zinc resources many eminent metallurgists are working on the problem of cheap and more efficient zinc reduction, and there can be no doubt that some process will be devised for local application which will make profit for both miner and metallurgist. It is also significant that Leadville mining men are proceeding at once in the search for suitable processes for low-grade ores without postponing such action to the day of reckoning which inevitably comes when the high-grade and easily treated ores have been exhausted. Although no definite announcement can be made, it is known that already negotiations are proceeding between the mine owners and metallurgists who believe that they can solve the problem.

#### Some Interesting Company Reports.

The Utah Copper Company has issued its report for the fourth quarter of 1910. The work of remodeling the Arthur plant to conform to the system adopted in the Magna plant is progressing satisfactorily, and the first remodeling unit will be placed in commission at once. The production of copper for the quarter was as follows:

October .....	7,582,219 lb.
November ..	7,468,915 lb.
December ..	7,137,585 lb.
Total.....	22,188,719 lb.
Average monthly production.....	7,396,239 lb.
Total production for 1910.....	89,019,511 lb.



The average net cost of copper produced for the quarter, after smelter deductions were made, was 8.227 cents per pound, as compared with 8.25 cents for the preceding quarter. The average grade of the ore was 1.56 per cent copper.

Of still more interest from the point of low cost of production of copper is the report of the Nevada Consolidated Copper Company for the fourth quarter of 1910. The output of copper for that quarter was but 15,698,595 lb., a decrease of about 15 per cent compared with the preceding quarter. The ore treated averaged 2 per cent copper, and the cost per pound of metal produced, after writing off depreciation on the Step-toe plant, is 6.7 cents. The total earnings of the company for the quarter were \$1,046,527.

The Goldfield Consolidated Mines Company's annual report for the fiscal year ending Oct. 31, 1910, contains some interesting facts bearing on gold recovery. The mill ore treated amounted to 265,351 tons, of an average value of \$36.99. The highest grade of mill ore was treated in April, 1910, when the value was \$51.06, and the lowest in February, 1910, the value being \$30.86. The average percentage extraction for the year was 93.98; the highest percentage was 95.78 in April and the lowest was 92.41 per cent in October, 1910.

Comparison with last year's report will show that there has been no decrease in operating costs coincident with increased tonnage at the 100-stamp mill. This is accounted for wholly by increased cyanide consumption, caused by the constantly diminishing proportion of oxidized ores and increase of sulphides from the lower levels. Leaving this factor out of consideration the plant has worked exceedingly well. In crushing efficiency, cost of operation and recovery of gold it has fully met expectations.

The capacity of the 100-stamp mill was increased during the year by the addition of six 6-ft. Trent Chilean mills and 24 No. 3 Deister slime tables. The installation has increased capacity from 650 tons to 850 tons per day and has been highly successful. If the cost for chemicals were the same this year as last the report of operations would show a reduction of 35 cents per ton for this year as compared with last. A new refinery is being built, located at some distance from the plant. It is of reinforced concrete construction and will cost about \$25,000. It will effect a saving of \$5,000 per month. A new and complete fire protection system has been installed since the fire last year. This cost approximately \$50,000, but will prevent the recurrence of such accidents as that of last year, which cost the company \$156,446.

#### The First Cyanamide Process Installation.

It is rather complimentary to the soundness of the principles of the Clancy cyanide process that within a few months after they were publicly announced a Colorado company decided to erect a mill embodying them. The Ajax Gold Mining Company, with mines at Victor, Cripple Creek mining district, Colorado, is erecting a 100-ton unit for the treatment of raw sulpho-tellurides by the Clancy process. The ore will be crushed dry, ground in tube mill with cyanide solution, agitated in tanks with the addition of potassium iodide and the application of the electric current; then, with the addition of calcium cyanamide, the solution will be regenerated in the same agitation tanks. The pulp will be filtered by the Moore system and the solution returned for further use.

Preliminary experiments in a laboratory way have thrown more light on the details of commercial operation. For one thing, it has been found that no such current densities as first proposed will be necessary for either extraction or regeneration. The extraction occurs with currents as low as 2 amp to 3 amp, and some experiments have indicated that still lower currents will produce the result desired. The regeneration of the solution also has been accomplished with similar currents and the same solution used for eight to ten weeks has shown little loss of strength. The proper substance to be used as anode is one that has not been settled, although the choice

probably rests between fused ferric oxide and Acheson graphite.

Construction of the plant is now under way and the first operation should take place early in the coming summer. The result will be watched for eagerly, as its success will mean much to the Western field. Present laboratory indications are very favorable for success in the commercial application.

#### The Government-Anaconda Smoke Suit.

If reports are to be credited, and they seem reliable, the federal government is to withdraw its suit against the Anaconda Copper Mining Company in which the plaintiff sought to compel the smelting company to abate the smoke nuisance at Anaconda, Mont. By such an action the government will be credited with acting sensibly in a suit which, in the opinion of many conversant with the facts and conditions, never could have been brought to a successful issue. Personal inspection on the part of government officials has evidently shown them that the Anaconda company has at all times acted fairly and that it proceeded wisely in the first instance by selecting a good place for the erection of the smelter and by making every reasonable provision for getting rid of fume and smoke to the best advantage of all concerned. A change of heart sometimes can be produced by getting the opponent's viewpoint and this seems to have been accomplished in the present instance.

#### Utah.

The milling interests of some sections of Utah are exercised over the recent introduction into the State Legislature of a bill making it a misdemeanor to discharge mill tailings into a stream. The bill is in the interests of fish preservation, but the mill operators see in it a sure means of closing their plants if it should carry. Inasmuch as but few streams of the State are polluted by mill tailings, and many others are left untouched, it is probable that a compromise can be effected and the matter settled without resort to legislation. The spirit of live and let live, and the greatest good for the greatest number, must prevail if all industries are to thrive.

#### The Iron and Steel Market,

Bookings of orders for finished steel products during February averaged between 60 per cent and 70 per cent of the total available capacity, being in excess of capacity in wire products, heavy in pipe, less heavy in bars, tin plates and sheets, and relatively light in plates, structural shapes and rails. During January there was a large improvement in the rate of booking and the rate of the closing days of January was maintained through February without important change.

Production has increased steadily through February on account of the increased rate of booking. At the close of the month the steel industry as a whole is running at between 65 per cent and 70 per cent of full capacity, against a trifle less than 50 per cent in the closing week of December.

The increased rate of buying of finished products is due chiefly to the fact that on account of uncertainties as to price and the desire to withhold business until after the annual inventory a considerable volume of business backed up and began to be released after the middle of January. There does not seem to have been any particular change in fundamental conditions, and an improvement in general business is therefore expected to effect an increase in orders from the rate prevailing in February.

The United States Steel Corporation on Feb. 10 announced its "unfilled orders" at the close of business Jan. 31 at 3,110,919 tons. This represented a gain of 436,162 tons during January, the first gain for a year. The total loss during 1910 was no less than 3,252,274 tons, the high point in unfilled orders since 1907 having been the 5,927,031 tons reported for Dec. 31, 1909. The capacity of the steel corporation, in terms of the products it sells, is commonly taken at about 1,200,000 tons monthly, of

which the January gain of 436,162 tons is 36 per cent, and its shipments in January may be estimated roughly at 40 per cent of capacity, with bookings at 75 per cent. The corporation's January bookings were particularly heavy by reason of the formal entering of a number of rail orders, which had practically been placed some time before. February has not witnessed such heavy booking of rail orders, and while in most products the month's bookings exceed those of January, the grand total may perhaps be found not as large.

The trend of finished steel prices has been upward, although not markedly so, the condition being important chiefly because it presents so strong a contrast to that supposed to exist in December, when there was no expectation of price changes, except in the downward direction. The wire advance of Jan. 23 and the tin-plate advance of Feb. 4 have served to show that finished steel prices can advance and while there is no general advancing tendency it has become patent that the price level is well held and that upon occasion advances can be effected.

The price of tin plates was advanced 10 cents per base box at a conference of producers held in Pittsburgh Feb. 3, the advance becoming effective at the opening of business Feb. 4. The new price basis is \$3.70 per base box for 100-lb. coke plates, with corresponding advances for ordinary ternes and heavy-coated ternes and bright plates. The advance was made solely to cover the increased cost of manufacture due to the advance in pig tin effected by the foreign syndicate. There is no means of forecasting the future of the pig tin market, but it may be taken as settled that should the price advance to 48 cents or 50 cents a further advance of 10 cents or 15 cents per box would be made in tin plate. When the former price of \$3.60 was established on tin plate, Nov. 12, 1909, pig tin was 30.50 cents and sheet bars \$28.50. When the advance to \$3.70 was made, Feb. 3, 1910, pig tin was 41.40 cents and sheet bars \$24.

All the prospects of the iron and steel industry are distinctly favorable. There has been improvement since the first of the year, and with occasional backsets improvement is expected through the remainder of this year. The causes of the continued decline in prices and activity in 1910 are readily perceived, first, that the recovery of 1909 was overdone, and, second, that during 1910 general trade conditions became less favorable. In its adjustment the iron trade swung too far and at the beginning of this year was in line for a return to normal.

#### Pig Iron.

The pig-iron market experienced a substantial, though slight, recovery in February. Prices had reached a level giving very small profits to most furnaces and involving to the Mahoning and Shenango Valley furnaces an actual loss. The fractional gains in February were important as showing a change in the price trend rather than as making a material difference in the general level. The International Harvester Company purchased a total of about 100,000 tons of foundry iron, divided between Northern and Southern, and one or two other Central Western consumers were large buyers. Southern iron stiffened so that concessions from the \$11, Birmingham, price on high-phosphorus irons disappeared and on standard Alabama grades the \$11 price became applicable only to deliveries this side July 1, later deliveries being held at 50 cents advance. The Buffalo market, after making prices of \$13.75, Buffalo, and perhaps lower, for No. 2 X, in competition with eastern Pennsylvania furnaces for New England business, advanced to \$14 and higher. Basic iron in eastern Pennsylvania advanced to \$14.50, delivered, after some large sales at \$14.25, other grades being unchanged in the East. In the Pittsburgh Valley market Bessemer remained at \$15, valley; \$15.90, Pittsburgh, as it has been since August, there being a number of sales of small tonnages, including one lot of 1000 tons for prompt delivery and another of 1250 tons for delivery 250 tons per month. The foundry-iron market firmed up, all prices at \$13.75, furnace, being withdrawn and leaving the market squarely on the

basis of \$14 at furnace, equal to \$14.75 to \$14.90, Pittsburgh, depending on location. A few small lots have been sold at \$14.25, furnace. Basic iron has furnished by far the most interest. Owing to the disappearance of one merchant furnace seller through the sale of the Midland properties to the Crucible Steel Company one or two other furnace interests marked up their price from \$13.25, valley, at which sales were made early in February, to \$14, furnace, which during the second half of the month was the lowest price made by furnaces. Meanwhile sales were made at advancing prices by middle interests, the course being pretty steadily that after one or two sales at a given price a slightly higher price would be demanded on the next inquiry. A sale of 10,000 tons was made at \$13.50, then sales of 5000 tons or more were made at \$13.60, this being followed by a sale of 2000 tons at \$13.65 and this in turn by several odd lots at \$13.70 and \$13.75, leaving the market in the closing days of the month on the basis of \$14 asked by furnaces, and last market sales at about \$13.75, equivalent to \$14.65 to \$14.90, Pittsburgh.

#### Steel.

The market has been very steady at \$23 for billets and \$24 for sheet bars, f.o.b. Pittsburgh or Youngstown, for either Bessemer or open-hearth, and \$29, Pittsburgh, for rods. A decided disinclination to sell open-hearth sheet bars at \$24 developed about the middle of the month. One interest, which had sold a special quality of open-hearth bars, suitable for deep stamping, at \$24, obtained \$25 on a lot of upwards of 2000 tons. Specifications on contracts for billets and sheet bars have improved and one branch of the consuming trade, the tin-plate mills, have been specifying practically their maximum tonnages on regular term contracts.

#### Finished Steel.

Finished steel prices are firmly held all along the line, as has been the case since early in December, and are as follows, f.o.b. Pittsburgh, except where otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Plates, 1.40 cents for tank quality.

Shapes, 1.40 cents for beams and channels, 3 in. to 15 in., inclusive, zeos and angles 2 x 3 and larger.

Steel bars, 1.40 cents, base; iron bars, 1.40 cents, Pittsburgh; 1.30 cents, Chicago.

Wire nails, \$1.75, base, per keg; plain wire, 1.55 cents, base; galvanized barb wire, 2.05 cents; painted barb wire, 1.75 cents.

Black sheets, 28 gage, 2.20 cents; galvanized, 3.20 cents; blue annealed, 10 gauge, 1.65 cents; painted corrugated roofing, \$1.55; galvanized, \$2.75 per square.

Tin plates, \$3.60 for 100-lb. cokes.

#### The Non-Ferrous Metal Market

Since our last report the metal market has been without feature except as to transactions in tin and copper. A fair volume of business continues to be transacted, but the uncertainties of the market confine purchases almost to immediate needs.

**Copper.**—The Copper Producers' Association report for January was made public on Feb. 8 and showed an increase in surplus. The production and imports for January were 115,676,591 lb. and although this figure showed a decrease of 7,642,628 lb. as compared with December the decrease in exports gave an ultimate surplus on Feb. 1 of 142,439,490 lb., which was an increase of 20,409,295 lb. This increase is not viewed unfavorably by a Western authority, who regards the increase as due to natural causes occurring at or near the end of the year when producers and refiners make a special effort to clean up accumulated stocks. The market has been unsettled and the agencies have effected business at a concession from the asking price. Lake copper is quoted at 12.5



cents to 12.75 cents, and electrolytic at 12.25 cents to 12.30 cents.

**Lead.**—The market has been quiet and without feature. The American Smelting & Refining Company has maintained a price of 4.50 cents, but other sellers have not reached this figure. New York quotation is 4.45 cents to 4.50 cents and St. Louis, 4.27½ cents to 4.30 cents.

**Tin.**—Since our last report tin has reached a high figure, selling at about 43 cents, but has dropped again quite as suddenly. The market has been spasmodic owing to the uncertainty of the intentions of those who control the foreign market. The last New York quotations are about 40 cents.

**Spelter.**—The market has been unsatisfactory, but a fair volume of business is reported and the prospect is for better conditions. The New York market quotes the metal at 5.42½ cents to 5.47½ cents; St. Louis at 5.27½ to 5.32½, which is lower than last month.

## CORRESPONDENCE

### Efficiencies of Electric Furnaces.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—In the last volume of the *Transactions* of the American Electrochemical Society a statement was published, apparently with deliberation, which, if left unanswered, might seriously mislead those who accept it.

The statement is that the lower the temperature of the outside of furnace walls of different materials the less the heat loss through the walls, hence the higher the efficiency of the insulation of the walls. The thickness of the walls is assumed to be the same, and for the purpose of comparing different materials composing the walls of this given thickness (as proposed by that author) it is assumed, for discussing the present feature, that the nature of the outside covering is the same. In other words, all conditions are the same except the material of the walls, the loss through them and the outside temperature.

It is furthermore stated by that author, by way of explanation, that the higher the external temperature the less the heat gradient from the inside to the outside and, therefore, the less heat is being taken out of the furnace. This evidently contradicts the first statement. The explanation is correct when applied to any one particular furnace—that is, to any one particular wall material; but it would certainly be incorrect to say that, in comparing different materials with each other, a smaller heat gradient necessarily means less loss. The loss is dependent not only on the heat gradient, but also quite as much on the conductivity of the particular material. Hence the outside temperature of the furnace is absolutely no criterion of the losses through walls of different materials, although that author claims very specifically that it is, whatever else he may have meant by his other remarks, which are not very clear.

The total fallacy of this statement and its alleged explanation is best shown by an actual example. Let us take the same small, simple furnace for which the calculation of the loss was given by me on page 15 of your January issue. The interior is 6 in. x 6 in. x 6 in. The wall thickness is 4 in., and for simplicity let us omit the second layer of brick. Let the temperature of the inside be 1500° C. and on the outside of the wall it will then be 175° when the wall is made of silica brick, which is assumed to have a thermal resistivity of 47 thermal ohms in cubic inch units. The loss through the walls will then be 3½ kw, as was there shown.

Now, for testing the truth of the above mentioned statement, let us assume the walls of the same furnace to be made of copper. This, of course, is an exaggerated case, but the results apply exactly the same, only to a less degree, to any other material. The thermal resistivity of copper is about 0.1 in cubic inch units. And let it be assumed that the outside

temperature now is half what it was before, hence 88° C. If the statement under discussion were correct the loss should now be considerably less because the heat gradient is much greater.

Calculations, however, made just as before show that the loss now is 1760 kw, or more than 500 times as great.

If the outside were allowed to get to a bright red heat, say to a temperature of 1000° C., nearly to the melting point of copper and therefore very much higher than the 175° for the brick, the loss will be 625 kw, or nearly 200 times as great, instead of being much less.

The above examples show that for either a higher or a lower outside temperature the loss may be enormously greater. By repeating the calculations for a very high resisting material it could similarly be shown that for both higher and somewhat lower outside temperatures the loss is less than in the original brick furnace. Hence the temperature alone can be no criterion whatsoever in such comparisons of different materials.

Hence the statement referred to is certainly incorrect; one would be committing very serious errors by relying on it. Different materials for walls, therefore, cannot be compared in this way, as that author had proposed. A smaller heat gradient is by no means an indication of a smaller flow of heat through the walls of different materials, but only for the same material. Such a statement should not have been left unchallenged at a meeting of the American Electrochemical Society. Too many incorrect statements have already been made concerning problems involving the flow of heat; as a matter of fact, such problems are generally extremely simple, when correctly understood.

Philadelphia, Pa.

CARL HERING.

### The Cyanamide Process in the Metallurgy of Gold.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—In your January, 1911, issue there appears an abstract of a paper presented before the Electrochemical Society, Dec. 16, 1910, on the "Clancy Process" of ore-treatment. In the paper referred to Mr. Clancy claims as novel the application of calcium cyanamide as a cheap cyanogen agent in the place of the more expensive sodium or potassium cyanide universally employed in the treatment of gold and silver ores by cyanidation.

I beg to take issue with Mr. Clancy as regards his claim to originality to employ calcium cyanamide in ore-treatment in place of the more expensive sodium or potassium cyanide as I can show priority in the application of this idea to the extraction of gold and silver from ores. And while Mr. Clancy may claim some novelty in adapting the use of calcium cyanamide to gold and silver extraction by means of electrolysis instead of the more direct method which I have proposed, viz: its transformation into calcium cyanide in a much more simple manner, by subjecting the cyanamide at a red heat to acetylene, water gas or any hydrocarbon gas rich in carbon. This can be simultaneously effected during the process of manufacturing the calcium cyanamide from calcium carbide.

I cannot understand how Mr. Clancy could possibly have presented calcium cyanamide as a cheap and novel substitute of his own invention in place of the more expensive KCy or Na Cy for use in the cyanide treatment of ores, when I already wrote up its application several years ago, first in: My United States patent 911,254, filed Sept. 4, 1907, and issued Feb. 2, 1909, from which I quote the following:

"My invention relates to the treatment of ores and tailing with ammonia-cyanide solutions, for the purpose of extracting their gold and silver content.

"My invention is applicable, not only to the treatment of ores and tailings containing gold and silver alone, but also to such ores and tailings as contain, in addition to gold and silver, such ammonia-soluble metals as copper, zinc, nickel and cobalt.



"The object of my invention is to effect economy in the process; and, to this end, my invention consists in the treatment of the ores and tailings with a solution of ammonia and alkaline earth-metal cyanide, as I shall now fully describe.

"In the treatment of ordinary gold and silver bearing ores and tailings by the cyanide process a cyanide solvent must be used which is not only stable in dilute solutions, such as 0.05 per cent to 0.50 per cent KCy, but must withstand exposure to the atmosphere as well, without decomposition. It must also not contain impurities, such as sulphide, carbides, etc., which would greatly depress the gold and silver extraction coefficient of the cyanide solvent. The cyanides ordinarily used, therefore, are the potassium and sodium salts, KCy and Na Cy.

"These are highly refined, snow-white and almost chemically pure. In this state they are sent to market for precious metal extraction purposes, and are, necessarily, very costly, from 20 to 25 cents per pound. In recent years, however, there has been great development in the cheap production of alkaline earth-metal cyanides, such as calcium cyanide,  $\text{Ca}(\text{CN})_2$  and Barium cyanide,  $\text{Ba}(\text{CN})_2$ , by means of the electric furnace, from atmospheric nitrogen.

"I have found that these electric furnace products, even impure as they are, if they contain 80 per cent to 90 per cent of the theoretical quantity of cyanogen  $(\text{CN})_2$ , are available for and may be adopted and used in ammonia solutions, as gold and silver solvents.

"In carrying out my invention, the crude calcium cyanide or barium cyanide is dissolved in ammonia solution of sufficient strength; the insoluble residue allowed to settle and the clear solution applied in the treatment of gold and silver bearing ores and tailings, not amenable to ordinary cyanide treatment, either on account of excessive cyanide consumption due to cyanicides, or on account of percentage of metals such as copper, nickel, zinc or cobalt, which render inert the gold or silver solvent powers of a dilute cyanide solution (KCy or NaCy) such as is commonly used, and cause an excessive loss of the cyanide.

"A working solution, according to my invention, may be made up as follows: Solution of ammonia, 1 per cent or less to 10 per cent  $(\text{NH}_3)$ . Calcium cyanide or barium cyanide, 0.05 per cent to 1 per cent. If this solution be too alkaline, a salt of ammonia, such as the sulphate, may be added to counteract such excessive alkalinity.

"The treatment of the ores or tailings with this solution is carried out in any of the usual or known manners, as by leaching or agitating them with such solution; and, where the ores or tailings contain, in addition to the gold and silver values, such metals as copper, nickel, zinc or cobalt, all these metals including the gold and silver are simultaneously extracted, and the separation of the gold and silver afterwards effected in any ordinary manner.

"It will be seen that by my improvement a very cheap gold and silver solvent is obtained, which overcomes all the objections and conditions above referred to.

"Having thus described my invention, what I claim as new and desire to secure by letters patent is:

"1. The improvement in treating ores and tailings containing gold and silver, which consists in extracting the precious metals with a solution of ammonia and an alkaline earth-metal cyanide.

"2. The improvement in treating copper, nickel, zinc and cobalt ores and tailings with gold and silver values, which consists in simultaneously extracting all the metals with a solution of ammonia and an alkaline earth-metal cyanide."

Then the following periodicals published articles and abstracts relative to the subject: *Mining and Scientific Press*, May 15, 1909; *Engineering and Mining Journal*, April 17, 1909 (Letter), D. Mosher; *Engineering and Mining Journal*, Feb. 27, 1909, (Letter), S. E. Bretherton; *Pacific Miner*, March, 1910; *Mexican Mining Journal*, August, 1910; *Metallurgical and*

*Chemical Engineering*, May, 1910 (Abstract); *Mineral Industry*, Vol. 18, 1909, page 365 (Abstract).

In the above articles, abstracts and letters, calcium cyanide as a cheap cyanide salt substitute produced from calcium cyanamide as per equation:



was several times noted and its use recommended by me as a cheap cyanogen agent for extracting gold and silver from their ores.

I trust I have made clear my position in regard to the "Clancy Process" as relates to the use of cyanamide. The fact that Mr. Clancy employs iodide in conjunction with the cyanamide is of no importance to the intrinsic claim involved of employing cheap cyanamide as a substitute for the expensive cyanides of sodium or potassium.

San Francisco, Cal.

D. MOSHER.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—After a careful perusal of Mr. Mosher's communication I fail to see any cause for discussion as to "priority" in the use of cyanamide in ore treatment. Nowhere, including his patent specifications, does it appear that he suggests the use of cyanamide per se as a solvent for the precious metals. On the other hand, he simply proposes, as many others before him have done, the use of calcium cyanamide as a starting point in the manufacture of calcium cyanide by heating to a red heat calcium cyanamide with other materials in an electric furnace and the product of this mixture—calcium cyanide—he proposes to apply to the treatment of ores.

The German chemists, Messrs. Frank and Caro, many years ago attempted to produce cyanides by this means, but found that this method did not work out. I am reliably informed by those actively interested in the manufacture of cyanamide that fortunes have been spent to make available the application of cyanamide in a thoroughly practical and economical way and these parties have very generously and frankly conceded that I have solved the problem. It seems to me that it would be more to the point if Mr. Mosher took issue with Frank and Caro for the determination of the question of "priority," for there appears to be little, if any, difference in the result finally accomplished by either of them, for they were apparently working along the same lines and the results quite naturally were the same.

As to the idea of my infringing any rights and prerogatives of Mr. Mosher, it will be apparent to anyone reading Mosher's specification that he does not use cyanamide in the cyanide solution, but proposes to use his electric furnace product, which contains calcium cyanide of about 80 per cent cyanogen content prepared by secondary electric furnace treatment of cyanamide, as already described.

In the Clancy process any amide or amidine compound may be used in conjunction with the cyanide solution, such, for example, as urea, guanidine, etc.

It is somewhat surprising that Mr. Mosher does not also claim priority to an amide or amidine compound in conjunction with the cyanide solution.

I trust that I have made the differentiation clear in regard to the Clancy process, as it relates to the use of cyanamide in the cyanide solution and not the use of the calcium cyanide product prepared from calcium cyanamide by special treatment as described by Mosher.

It is abundantly clear from reading Mr. Mosher's specification that he does not attempt to claim the use of raw cyanamide in the ore treatment process, for the simple reason that he, like many others, was not aware of the fact that cyanamide, amide or amidine compound could be used in conjunction with the cyanide solutions until the publication of my patent specifications and the articles which I have written disclosing the same.

New York City.

JOHN COLLINS CLANCY.

## A New Process for the Refining of Pig Iron.

By J. B. NAU.

The process to be discussed in this article has for its purpose the preliminary refining of pig iron, more especially of phosphoretic pig iron too high in silicon to be treated directly and economically in the open-hearth furnace.

Other methods with the same purpose in view are in successful operation to-day. Thus, for instance, the so-called duplex process, which, I believe, was first recommended by me in 1890 as the best method to treat the highly siliceous and phosphoretic pig iron of the South. As is well known the duplex process, in principle, consists in submitting the raw liquid pig iron to a blow in the Bessemer converter, whereby the silicon and part of the carbon are eliminated from the iron before the latter is taken to the open-hearth furnace.

Naturally in the Bessemer blow some iron is burnt besides the manganese, but no phosphorus is touched. The resulting metal will, therefore, still contain all its phosphorus and whatever carbon it is deemed necessary to leave unburnt in order to keep the metal fluid enough for subsequent operations. The blown metal being of a uniform and regular composition the process has become quite popular in spite of the loss of iron entailing the blow.

It was principally to avoid this loss of metal, as well as to remove phosphorus instead of carbon in the preliminary operation, that the present method was devised.

Several years ago I obtained a United States patent (786,048, March 28, 1905) on a refining method in which I proposed to submit pig iron, running from the blast furnace, to the refining action of solid pieces of iron ore. This method consists in running the liquid iron through a column of heated ore and let a bath of metal accumulate on the bottom of the apparatus in such a manner that the lower part of the refining ore column is kept immersed in the liquid iron.

The intimate contact between iron and ore at a high temperature will rapidly effect the refining to the desired extent and if the final temperature remains high enough both iron and resulting slag will remain fluid enough to be run easily and without difficulty from the refining apparatus.

The final temperature in the operation will be influenced by:

(1) The surplus or available heat stored away in the liquid iron running from the blast furnace. This temperature is mostly several hundred degrees Centigrade higher than necessary for good fluidity and the surplus heat thus carried can be made to largely contribute to the success of the refining operation.

(2) The heat brought into or developed by the reaction taking place in the refining. This heat depends on several factors, such as amount of gangue in the refining ore, sensible heat carried by the refining ore, which can easily be heated up to 1000° C. or more without melting.

It also depends on the amount of silicon, phosphorus, etc., present in the original pig iron.

The elimination of silicon by means of  $\text{Fe}_2\text{O}_3$  will be accompanied by a rise of temperature, while the elimination of phosphorus, and especially carbon, will under the same conditions be accompanied by an absorption of heat and consequently a lowering of temperature.

Hence a pig iron with a higher silicon content will be more satisfactory from a calorific standpoint.

Thus with a good temperature of raw pig iron and refining ore and satisfactory chemical composition of the same a final temperature can be obtained high enough for all practical purposes.

But a metallurgical process cannot be declared successful without its being so under all reasonable conditions. There would certainly occur many instances where the preliminary conditions necessary for success could not be obtained readily and where, therefore, the final temperature could not be kept high enough to have the metal sufficiently fluid.

The special purpose of the present invention is to provide the method patented by me several years ago with means that will make it successful at all times and under all reasonable conditions.

This can be best obtained by carrying out the operation in an electric furnace, especially built to create easy immersion of the refining ore in the metal to be refined.

### Outline Description of Electric Furnace.

The furnace used for the process may vary.

In the present instance I have devised an electrical furnace capped by a water-cooled C. I. roof and surmounted by an ore shaft with a lateral inlet for the liquid pig iron to be refined. Vertical electrodes penetrating the roof are placed around the shaft. The bottom is made of carbon paste and the electric current playing between the electrodes and the carbon bottom traverses the liquid metal and slag, thereby keeping it heated up to the desired temperature in case there is not enough heat developed by the reaction.

If the heat conditions are enough no electric current need be turned on. Thus success is made possible under all conditions.

The furnace is so arranged and the operation will be so conducted that the refining ore necessary will always be kept immersed or made to be immersed to any desired depth in the metal to be treated and as soon as the ore in the bath is consumed and has to be replaced by new ore the latter will always be made without any trouble to be immersed in the liquid bath. The means employed for doing this are of the simplest kind and the buoyancy of the liquid metal can never prevent the much lighter ore from sinking down into the heavier liquid metal wherever the special means employed for that purpose are put in operation.

As it will be shown hereafter that on the other hand perfect fluidity of the refined metal and the resulting slag will at all times be maintained through electrical heating the method cannot but be successful.

### Balance of Materials in the Elimination of Silicon, Phosphorus and Manganese.

In an article published in the *Iron Age* of March 23 and April 6, 1905, I have established a thermal calculation of the process which, owing to a mistake in the balance of material, I have since found to be not quite correct.

Therefore, in this paper I again establish that calculation and base it on a temperature of 1400° C. for the inflowing liquid iron from the blast furnace and 800° C. for the refining ore and guided by this thermal calculation I figure out how much electrical power I will require to keep the final temperature of the refined metal and the resulting slag at 1400° C.

I also calculate more roughly how much electrical power I will require when availing myself of the heat stored away in the pig iron to be refined to the extent of allowing the temperature of the refined metal to drop from 1400° C. to 1250° C. The calculations follow here in kilograms and Centigrades. Assumed composition of pig iron to be refined:

	Si	Ph	Mn	C
In percentage .....	2%	1%	1.5%	3.5%
Per ton .....	20 kg	10 kg	15 kg	35 kg
Removed in the refining.....	19 kg	8 kg	13 kg	0
Refined metal will contain.....	1 kg	2 kg	2 kg	35 kg
In percentage .....	0.1%	0.2%	0.2%	3.5%

The refining ore is supposed to contain 90 per cent of  $\text{Fe}_2\text{O}_3$  (63 per cent Fe) and 10 of gangue supposed to be  $\text{SiO}_2$ .

Slag to be formed is supposed to be in the nature of a puddle cinder with 50 per cent of metallic iron.

Results obtained in the refining:

19 kg of Si yielding 40.72 kg of $\text{SiO}_2$ will require 21.72 kg O	
8 kg of Ph yielding 18.32 kg of $\text{P}_2\text{O}_5$ will require 10.32 kg O	
12 kg of Mn yielding 16.78 kg of MnO will require 3.78 kg O	
40 kg eliminated	75.82 in slag
	35.82 kg O

This O being solely derived from the refining ore the amount of the latter must be sufficient

(1) To oxidize the elements Si, P and Mn to the extent indicated;

(2) To obtain a slag containing 50 per cent of metallic iron, corresponding to 64.283 per cent of FeO.

The unknown quantities in our problem are:

The weight of the slag obtained..... =  $x$

The weight of the ore used..... =  $y$

We know that the ore used contains 10 per cent of  $\text{SiO}_2$  and (90 per cent of  $\text{Fe}_2\text{O}_3$  = 63 per cent Fe and 27 per cent O), hence  $y = 0.63 y$  iron +  $0.27 y$  O +  $0.10 y$   $\text{SiO}_2$ .

We also assume that the slag contains 64.283 per cent FeO and 35.714 per cent gangue = 100 per cent, hence  $x = 0.25714 \times \text{gangue} = 0.64286 x$  FeO.

The total oxygen required in the refining is 35.82 kg and the weight of the oxidized elements removed is 75.82 kg.

As the gangue of the slag is composed of the gangue derived from the ore and the oxidized elements eliminated from the iron we have equations

$$(1) \quad 0.35714 x = 75.82 + 0.10 y$$

On the other hand the total oxygen will be derived from the ore, hence equation

$$(2) \quad 35.82 + \frac{16}{72} \times 0.64286 x = 0.27 y$$

The two equations will finally give us:

$y = 287.6$  kg of ore and

$x = 292.83$  of slag = 104.582 kg gangue and 188.248 Fe O (50 per cent Fe).

The 287.6 kg ore containing 90 per cent = 258.85 kg of

$\text{Fe}_2\text{O}_3$  with..... 77.652  
kg O, of which will be used for purifying..... 35.820

Leaving for FeO in slag..... 41.832

On the other hand, the slag contains 50 per cent = 146.415 Fe, corresponding to  $146.415 \times 2/7 = 41.833$  kg O, obtained from 209.164  $\text{Fe}_2\text{O}_3$ .

This leaves  $258.85 - 209.16 = 49.69$   $\text{Fe}_2\text{O}_3$ , yielding 34.77 kg directly reduced iron.

Composition of resulting slag:

$\text{SiO}_2$ derived from ore $287.6 \times 0.10 = 28.76$	In %
$\text{SiO}_2$ derived from refining = 40.72	69.48..... 23.72
$\text{P}_2\text{O}_5$ ..... 18.32 kg	6.26%
MnO ..... 16.78	5.73
FeO ..... 188.25	64.286
	99.996%

The balance sheet of materials will be:

Put in Refiner.	Taken Out of Refiner.
Pig iron ..... 1000	Refined iron ..... 960
Iron ore ..... 287.6	Slag ..... 292.83
	Iron reduced ..... 34.77
Total ..... 1287.6	Total ..... 1287.60

Thus, under the conditions considered, each 1000 kg of pig iron treated will yield: 994.77 kg of refined iron containing very nearly 0.1 per cent of Si—0.2 per cent Ph—0.2 per cent Mn and 3.5 C and 292.83 kg of slag with 50 per cent of Fe.

No addition of flux, such as, for instance, limestone, was considered. The presence of limestone, on one hand, would no doubt reduce the amount of Fe found in the slag and, on the other hand, it might make it of considerably less value than a slag containing 50 per cent of iron and 5.73 per cent MnO.

#### Heat Balance in the Elimination of Silicon, Phosphorus and Manganese.

Specific heat of ore containing 10 per cent  $\text{SiO}_2$  and 90 per cent  $\text{Fe}_2\text{O}_3$  from  $0^\circ \text{C}$ . to  $800^\circ \text{C}$ . = ..... 0.21  
Specific heat of iron from  $0^\circ \text{C}$ . to  $800^\circ \text{C}$ . ..... 0.157  
Specific heat of slag from  $0^\circ \text{C}$ . to  $800^\circ \text{C}$ . ..... 0.21  
Total heat carried by slag at  $1400^\circ \text{C}$ . ..... 500 calories

Total heat carried by liquid iron at $1400^\circ \text{C}$ . ....	300 calories
Silicon burning to $\text{SiO}_2$ develops.....	7830 "
Phosphorus burning to $\text{P}_2\text{O}_5$ develops.....	5760 "
Manganese burning to MnO develops.....	1723 "
Iron burning to $\text{Fe}_2\text{O}_3$ develops.....	1796 "
Iron burning to FeO develops.....	1352 "

The heat of formation of slag is not known, but considering that wherever heats of formation of phosphates and silicates were established the heats of formation of phosphates per unit of acid were always found to be much greater than the heats of formation of silicates uniting with the same amount of the same base one, no doubt, will remain far below the truth when taking for the phosphates the same heats of formation per unit of acid as for the corresponding silicates.

It may also be noticed that heat of formation per unit of a silicate of lime of lower basicity is considerably less than the heat of formation of a silicate of higher basicity, hence per analogy one may assume that the heat of formation per unit of acid of a silicate of iron of higher basicity is much higher than the heat of formation of FeO,  $\text{SiO}_2$ , the only one that to my knowledge was ever established.

Taking, therefore, the same heat of formation for phosphates and silicates and adopting as unit 150 calories obtained when one unit of  $\text{SiO}_2$  combines with FeO to form FeO,  $\text{SiO}_2$ , I undoubtedly will remain far below the truth and therefore remain within very conservative limits.

The heats of formation of silicides, phosphides and carbides are too uncertain or too little known to be considered here. Their dissociation would no doubt be accompanied by an absorption of heat and a lowering of temperature, but it may be fairly assumed that their omission will be counterbalanced by the omission of the source of heat resulting undoubtedly from the higher heat of formation of the slag than the one considered.

The heat lost by radiation was based on the figure of 440,000 calories per ton of iron established by Gruner in his studies of the blast furnace. As Gruner obtained this loss by difference between the total heat supplied and the actual heat absorbed, both in reactions and in the gases leaving the furnace, and as his calculations were based on a run of 24 hours of a blast furnace making less than 40 tons of iron per day—and as his loss includes also the heat carried off by the cooling water—the item of heat loss due to radiation in a refining apparatus of a capacity of 500 tons per day, having no cooling water and with very much less radiating surface than Gruner's blast furnace, may be considered very ample.

The itemized heat balance per ton of iron treated will follow here:

LIQUID IRON AT $1400^\circ \text{C}$ . <sup>1</sup> ORE AT $800^\circ \text{C}$ .	
Heat Supplied:	Calories.
19 kg of Si will give $19 \times 7830$ .....	148,770
8 kg of Ph will give $8 \times 5760$ .....	46,080
13 kg of Mn will give $13 \times 1723$ .....	22,399
59.04 $\text{SiO}_2$ and $\text{P}_2\text{O}_5$ will give $59.04 \times 150$ ....	8,854
$\text{SiO}_2$ from ore is omitted.	226,103
Heat Spent:	
35 kg iron—reduced from $\text{Fe}_2\text{O}_3 = 35 \times 1796 =$	62,860
heated from $800^\circ$ to $1400^\circ = 35$	
$(300 - 800 \times 0.157) =$ .....	6,104
293 — 40 = 253 kg of slag	
Heated from $800^\circ$ to $1400^\circ = 253$	
$(500 - 800 \times 0.21)$ .....	83,996
The 40 kg of constituents removed were at $1400^\circ$	
209.16 kg of $\text{Fe}_2\text{O}_3$ reduced to FeO	
$209.16 \times 444 =$ .....	92,867
Heat lost by radiation (per ton).....	25,000
	270,827
Heat deficiency .....	— 44,724



**Balance of Material in the Elimination of Carbon.**

All conditions remaining the same the above heat balance might be considered a constant.

So far we have not considered the carbon, whose elimination will largely depend on the temperature and the length of contact. Still the elimination of a small percentage only of carbon as CO will constitute a great disturbing factor in the heat balance and it is, therefore, necessary to consider its elimination within the limits that it can reasonably be expected to take place.

We apply the calculations to the elimination of 1 kg of C. We shall have again:

Ore necessary:  $y = 0.63 \text{ y iron} + 0.27 \text{ y O} + 0.10 \text{ y SiO}_2$   
Slag:  $x = 0.35714 \text{ x gangue} + 0.64286 \text{ x FeO}$

With the removal of carbon the weight of the slag is increased only by a corresponding amount of gangue and FeO (50 per cent Fe) derived from the ore.

Hence in this case we have

$$(1) \quad 0.35714 x = 0.10 y$$

The total oxygen entering into the reactions will be composed of  $\frac{4}{3}$  kg necessary to remove C as CO plus the oxygen of the FeO of the slag. Hence,

$$(2) \quad \frac{4}{3} + \frac{16}{72} \times 0.64286 x = 0.27 y$$

which equation will give us

Slag  $x = 1.623$  kg with 0.579 gangue and 1.044 FeO  
with ..... 0.232 O

Ore  $y = 5.7964$  kg with 0.579 gangue and  
5.218 kg of  $\text{Fe}_2\text{O}_3$  having 3.6526 iron and.. 1.5654 O  
Oxygen necessary to remove 1 C..... 1.3333

Leaving O in  $\text{Fe}_2\text{O}_3$  of slag..... 0.2321

Metallic iron reduced:  $3.6525 - 0.812 = 2.8406$  kg.

In regard to 1 kg of C the balance sheet of materials will therefore be:

Put in Refiner.		Taken Out.	
Ore = .....	5.796	Metallic iron reduced.	2.8406
Carbon = .....	1.000	Slag .....	1.623
		CO escaping .....	2.333
Total .....	6.796	Total .....	6.796

**COMPOSITION OF SLAG DUE TO THE ELIMINATION OF 1 KG OF C.**

	In %
$\text{SiO}_2 = \dots\dots\dots 0.579$	35.68
$\text{FeO} = \dots\dots\dots 1.044$	64.32 (50% Fe)
Total .....	1.623 kg 100%

**HEAT BALANCE DUE TO THE ELIMINATION OF 1 KG OF CARBON.**

Heat Giving Factors:	Calories.
1 kg of C burning to CO.....	2403
5.79 kg ore at $800^\circ - 5.79 \times 800 \times 0.21 = \dots$	972
1 kg of carbon at $1400^\circ \text{ C.} = \dots\dots\dots$	300
0.579 of $\text{SiO}_2$ uniting with FeO, 150 calories = ..	87
Total .....	+ 3762
Heat Consumed:	
2.84 of Fe reduced from $\text{Fe}_2\text{O}_3 = 2.84 \times 1796 =$	5100
2.84 of Fe heated from $800^\circ$ to $1400^\circ$ and melted = $2.84 (300 - 800 \times 0.17) = \dots\dots$	465
1.16 of $\text{Fe}_2\text{O}_3$ reduced to FeO = $1.16 \times 444 = \dots$	515
1.623 of slag heated from $800^\circ$ to $1400^\circ \text{ C.} =$	
$1.623 (500 - 800 \times 0.21) = \dots\dots\dots$	538
Heat carried off by 2.333 kg of CO at $500^\circ =$	
$\frac{(2.333}{1.25} = 1.8664 \text{ m.}^3) \times (0.303 + 0.0027 \times 5)$	
$\times 500 = \dots\dots\dots$	294
	- 6925

Deficiency of heat per kg of carbon eliminated = - 3163  
Considering that the whole operation necessary to treat one

ton of iron will not last more than 10 minutes, which is actually more than there is necessary in the wash process where hardly any carbon at all is removed, at a higher temperature of refining ore we suppose the carbon reduced from 3.50 per cent to 3.30 per cent corresponding to a total of 2 kg of carbon per ton of iron, the loss of heat due to this cause will be:

$$2 \times 3163 = 6326 \text{ calories}$$

Thus per ton of iron treated the whole loss of heat will be:

Calories.

Removal of silicon, phosphorus, manganese to the extent shown .....	44,724
Elimination of 2 kg of carbon.....	6,326

Total loss of heat per ton of metal treated..... 51,050

**Electrical Power Necessary.**

We propose to supply the above deficiency of heat by electrical power so as to keep the temperature of the refined metal and the resulting slag at  $1400^\circ \text{ C.}$

If we apply the method to the refining of all the iron made in 24 hours in a 500-ton furnace we will have to refine per hour  $500 \div 24 = 21$  tons and we will have to supply  $21 \times 51,050 = 1,072,050$  calories per hour.

Taking the efficiency of an electric furnace as 80 per cent the current would have to supply

$$\frac{1,072,050}{0.80} = 1,340,062 \text{ calories}$$

corresponding to

$$\frac{1,340,062}{640} = 2091, \text{ say, } 2100 \text{ electric hp-hours.}$$

This electrical horse-power would be enough to refine the iron as fast as the furnace can make it.

The calculations are based on heat factors as they are known to-day and on that basis the results obtained should be correct.

Still, some of them are to a certain extent doubtful and their use may influence the results and show the heat conditions to be different from what they might be found in practice.

There will be no harm if practice shows a smaller heat deficiency than the one established; but, on the other hand, the heat deficiency might be found larger.

When it is considered, however, that the final temperature of  $1400^\circ \text{ C.}$  is far above the temperature that perfectly fluid metal requires to easily run from the refining furnace, we have at our disposal a factor of safety of great value of which we have made no use. The temperature of the metal and cinder in a puddling furnace reaches hardly if ever  $1250^\circ$ . The temperature of pig iron running from the cupola seldom reaches  $1250^\circ \text{ C.}$  and is mostly below  $1200^\circ$ . Yet at these temperatures both puddle-cinder and cupola metal are very hot and very fluid.

It would therefore appear to be unnecessary to have a final temperature of  $1400^\circ$ . If we allow this temperature to drop to  $1250^\circ$  excellent results are assured and the refining operation will become far more economical in so far that now we actually make use of the surplus heat unnecessarily stored away in the liquid iron and a correspondingly large amount of heat to be supplied by electricity will be saved. A cursory examination of the problem made under this new aspect will give the following results:

Allowing the temperature to drop from  $1400^\circ \text{ C.}$  to  $1250^\circ \text{ C.}$  would liberate an amount of heat as shown below:

1000 kg of iron from 1400 to 1250 liberates $1000 \times$	
$150 \times 0.23 = \dots\dots\dots$	34,500 C
293 kg of slag liberates $293 \times 150 \times 0.30 = \dots\dots$	13,185 C

Total loss ..... 47,685 C

Leaving out all other factors that would make the case appear under a still more favorable light the amount of heat to be furnished by electricity will now be reduced to  $51,050 - 47,685 = 3365$  calories per ton, or per hour  $3365 \times 21 = 70,665$  calo-

ries, requiring a horse-power of  $\frac{70,665}{0.8 \times 640} = 138$  hp.

Since, roughly speaking, the 47,685 calories correspond to 150° C., 51,050 calories would correspond to about 160°, which means that no electricity would be required when allowing the temperature to drop to 1240°, high temperature of very fluid metal running from the cupola.

Thus at times when conditions are quite favorable no electricity need be applied for success; but with less favorable conditions some electricity would be necessary.

Probably at no time the full amount of 2100 hp will be required, but with the electrical power at hand always ready to be applied success is assured under all conditions.

It might further be stated that the size of the hearth or crucible of the furnace can be determined so as to keep the metal to be refined and the refining ore in contact for any reasonable length of time, thereby making it possible to extend the heating of the metal over a greater length of time, which naturally would afford every facility to again sufficiently raise the temperature of the outflowing metal should this latter have cooled down more than desirable.

New York City.

### Some Electric Furnace Notes.

By JOH HÄRDÉN, E. E.

The valuable contributions to the knowledge of the proper dimensions of furnace electrodes made by C. Hering in several issues of this paper will no doubt be greeted by furnace designers with acclamation. Many were the difficulties confronting the designer in the earlier days of the electric furnace era, both regarding the best size of electrodes and the most efficient way of arranging them, and also, last but not least, where to get them.

Some twelve years ago the author was given the problem of designing a graphitizing furnace for carbon rods in Russia.

The current available was direct current, about 3000 amps by 65 volts, no alternating current being obtainable at the works; in fact, the furnace was to be operated in series with a row of electrolytic cells and intended to absorb surplus power in a useful way. The carbon rods to be graphitized were about 3½ in. diameter by 4 ft. long. Owing to these two facts the operation could not be carried out "in bulk" as is the practice at Niagara, but had to be in a continuous operation. It was found subsequently that the best method under the circumstances was to pass three rods at a time slowly through the furnace, followed by three others in succession.

At first, attempts were made to produce a large arc in a carbonaceous atmosphere. This method had, for obvious reasons, to be abandoned very soon. The arc behaved very well for 6 to 8 hours, but as soon as the interior of the furnace got well heated the furnace walls became part of its path being conductive at that temperature, and even directing magnets had no beneficial influence owing to the irregular paths taken by the arc. Also, the rods to be treated got scorched and injured by the action of the arc.

In this connection it might be mentioned that the suggestion made by A. Mahlke in Vol. IX, No. 1, p. 42, to broaden the active ends of the electrodes, so as to obtain a better distribution of the arcs and the heat in the furnace, is not likely to be successful. Anyone who has had an opportunity to observe the arc, even in a furnace of the largest size, will have noticed how the base of the arc is always concentrated to a comparatively small area, wandering about on the surface of the electrode, how large this also may be in cross section. This is no doubt due to the fact that the arc requires carbon vapor of high temperature (or any other ionized medium of high temperature), and as this cannot take place at the same instant over the whole surface of the electrode, the arc will always be concentrated at its base. The same may also be

observed in a Mercury-vapor lamp; the arc forms only a comparatively small spot on the surface of the mercury cathode.

Reverting again to the graphitizing furnace a further attempt was made to convert it into a resistance furnace in the following way: Two electrodes, about 6½ in. square, were placed inclined on opposite sides into the furnace chamber of fire brick; the bottom consisted of electrode stumps and stamped coke. The furnace was partly filled with granular charcoal, in which the electrodes were embedded about 7 in. to 8 in.; due provision was taken to prevent the current passing through the furnace walls, similar to the arrangement used in the pig iron smelting furnace by Gronwall & Stalhane.

This arrangement would work satisfactorily for about a day or so; after that period, the trouble began. In spite of the air being excluded as far as possible from the electrode collars the electrodes were worn out just inside the furnace room very rapidly, while the active ends grew in thickness by the setting-on of carbonaceous matter to such an extent that the electrode took the shape of an irregular mushroom, causing great difficulties in the regulating.

A final alteration was then made, which proved successful. The bottom of the furnace was built up on a hollow cast-iron plate, similar to that in a carbide furnace, two of which were in operation at the works as well. Electrode stumps, about 8 in. high, were placed in carbon cement on the iron plate and the brickwork erected to form the furnace chamber.

This bottom plate formed the positive electrode, while the negative one consisted of a 6½-in. x 6½-in. carbon suspended from above, by means of a water cooled holder, through an opening in the furnace roof. The furnace chamber was filled with granular charcoal as before, in such a manner as to reach the upper electrode and sloping down at the sides, which prevented the current passing through the walls. The rods to be graphitized were passed three by three slowly through the glowing mass of charcoal and allowed to cool in coal dust on a gantry. The rods were examined as to degree of graphitizing by tapping with a steel bar; the sound emitted gave a good idea of the process; also the polishing effect by rubbing.

One difficulty remained to be solved; the roof was soon destroyed in each case, and we may assume that this difficulty is still a great stumbling block, even to-day, in all electrode furnaces. In the above instance, the problem was solved in this way: The roof of the furnace was arched in by means of scrapped or curved carbon electrodes, separated and wedged in by means of intermediate wedge-shaped high-grade fire bricks, to prevent leakage of current. After some adjustment this kind of roof could be made to last for 8 to 9 months in spite of the continuous run day and night at a tremendous temperature, and despite the fact that the furnace was emptied and fresh charcoal refilled every twenty-four hours, which caused a heavy fall in temperature in fact, this refilling was the primary cause of the rapid breakdown of the original brick roof. One top electrode would, under these circumstances, last four to five days, its dimensions being, as already stated, 6½ in. x 6½ in. x 4 ft. 6 in., and the current 3000 amps to 3500 amps at 40 volts to 45 volts. As far as is known to the author the bottom needed no repair at all during a campaign of four years.

A curious incident in the case of the trials with the arc-design was the upshot of a number of further experiments and also some patents.

When the large arc was started, violent rushes of current took place, the whole furnace being cool. This was strongly objected to by the power station master, as it caused very strong surging between the large engines running parallel; the regulating device of the steam engines was of such nature as to be particularly sensitive in this respect.

While awaiting the construction of a suitable starting resistance (an iron band immersed in running water) a makeshift was made in trying to start the arc, and also to prevent



its extinguishing, by means of a spark coil, made to discharge between the electrodes. In order to prevent the spark taking any other way than was intended, a condenser was connected to the sparking circuit and a primitive reaction in the power circuit.

By this means it was quite possible to start the arc without any serious current rush and the tendency of the arc to go out before the furnace was heated ceased if the spark was kept going.

This device was first tested with a smaller arc and narrower electrodes; it was then observed that the tips of the electrodes got more quickly graphitized and that the graphite formed was of a more flaky appearance when the spark had been in operation for some time, though not always, only under certain conditions. The whole phenomenon was at first thought to be accidental only, but on further investigation it was found that the graphitizing could, under favorable conditions, be somewhat accelerated by introducing condenser discharges of practical magnitude. A specially made spark coil and a suitable condenser were therefore installed and put into permanent use; the workmen were protected from the discharge by means of heavy mica collars, surrounding the auxiliary electrodes through which the discharges were introduced into the furnaces.

Other and more pressing work prevented a more systematic and exhaustive investigation as to the economical efficiency of the method; it may be said in this respect, however, that its use appeared to be justified. The workers, a man and a boy per shift, were paid after piecework for a minimum number of rods, passed by the foreman after each shift, and an additional bonus for each extra rod passed. It was regularly found that if the spark coil had been out of operation for some

found that this did not answer. The pins were rapidly worn out, regardless of what metal they were made; besides, the mercury soon got clogged up in spite of a thick strata of oil over it. Carbon rods lasted longer, but the spark was greatly diminished, owing to insufficient rapidity of break.

A turbo-interrupter was then tried, in which a mercury jet in the well-known manner was projected against a toothed iron wheel, moving in oil or cooling liquid. Even this proved futile, as the teeth of the wheel were soon worn off by the sparking; the mercury got so clogged in three or four days that it had to be replaced. To prevent this latter nuisance, a jet of coal-gas was introduced into the vessel, so as to prevent oxidation, but even this measure, though it did some

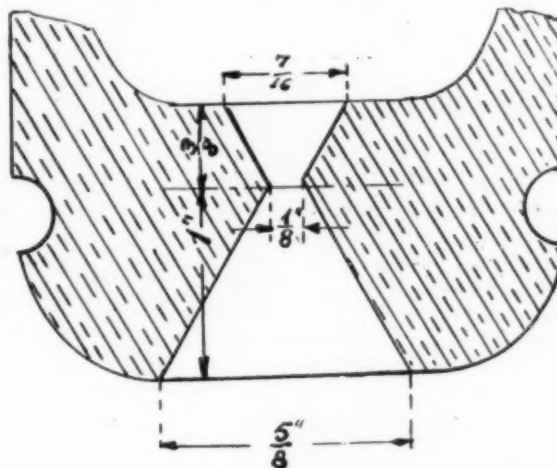


FIG. 2.—CORRECT SHAPE OF BORING.

good, did not secure from breakdowns to the extent of commercial efficiency.

The electrolytic interrupter was finally resorted to and, in spite of its low electric efficiency, proved to be more satisfactory than the others. A platinum point, as introduced by Dr. Wehnelt, was first tried, but was soon destroyed by the current. After a good deal of experimenting it was found to answer the purpose best if a large porcelain insulator drilled through the top, without any metal point at all, was used.

As indicated in Figure 1, the insulator was suspended in a doubled-walled vessel of lead, cooled by running water. A lead spiral formed the inner, or positive electrode; the active liquid was sulphuric acid of about 1.14 Sp.W. The paper condenser in the base of the coil was not in circuit while this interrupter was at work.

With this device it was found quite possible to keep the coil going day and night for many months, only stopping about every fifth day for cleaning and replacing of the acid, which was gradually vaporized; but such stoppage only required about 15 to 20 minutes.

The hole drilled in the insulator was gradually widened by the action of the current. It had to be replaced about every three months, for which purpose a spare one was always kept.

At the beginning, some difficulty was found in the tendency of the apparatus to act as a pump. In some cases the acid was driven out through the bottom hole into the outer vessel, causing the working to be irregular and producing much heat. Again, with another insulator substituted, the acid rose in the inner room so as to overflow and cause short circuit.

It was soon found that the shape of the hole was responsible for this behavior. If the hole sloped too much upwards, the gas bubble causing the interruption exploded upwards, thus driving the pile of liquid up into the inner vessel, causing a surging from underneath, so that the liquid in the outer vessel rushed inwards. If this happened rhythmically, as was often the case, the insulator was soon filled to overflowing.

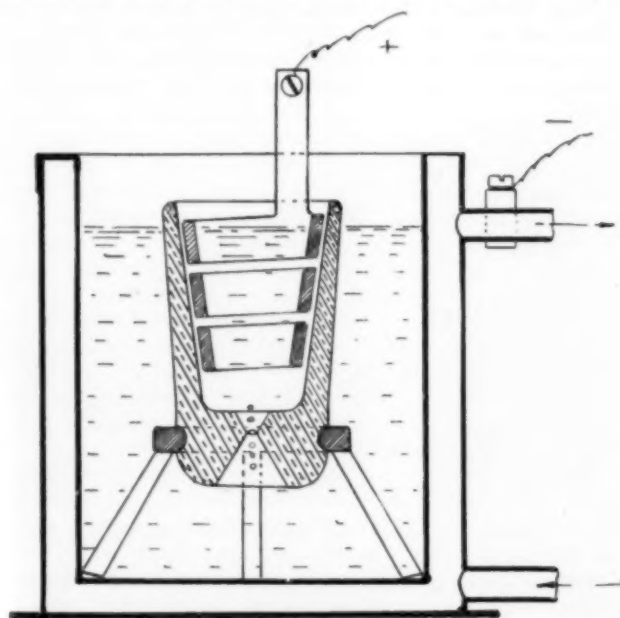


FIG. 1.—INTERRUPTER DESIGN.

time during a shift the number of rods passing examination decreased and the rejected rods had to be sent through the furnace again; therefore the workmen showed a keen interest in keeping the coil in going order.

As there was no alternating current available at the works, the coil had perforce to be operated by direct current, 125 volts. Anyone who has had the task of keeping a large coil, giving a 12-in. to 14-in. spark of great volume, going day and night without breakdown for several months at a time will testify that the finding of a suitable interrupter is by no means an easy problem.

At first, the ordinary mercury make and break, consisting of two motor-driven vertical pins, was tried, but it was soon



On the contrary, if the boring widened too acutely *downwards* the bubbles burst outwards and drove the liquid into the outer vessel.

In Fig. 2 is indicated the correct shape of the boring, with which no surging took place. The hole should be well polished, or preferably glazed, when the action was much minimized. The period of interruption was adjusted by means of a variable inductance in the primary circuit, so as to give the best effect.

Large glass condensers were used in the secondary circuit to produce the oscillations necessary.

With this arrangement the breakdowns were very rare, and the whole apparatus could be left for four to five days without inspection.

London, England.

### Metallurgical Engineering Education.

By VICTOR C. ALDERSON.

The forces which influence the course of study in our technical schools are different from those which influence the cultural college. In the case of the latter these influences are from within. The members of the faculty offer such courses as they deem wise, treat them as they choose, and develop such a man as seems good to them. The graduate goes out trained for nothing in particular; if he succeeds in business, for example, his success is due rather to his natural aptitude for business than to any special training the college of general culture has given him.

In the case of the technical school the influences which determine the training a man shall receive come from without. He enters a profession and must understand its fundamental principles and must start at the bottom. The standard for graduation—the level he must attain—is determined by the profession he intends to enter.

For this reason, the work of our technical schools is more exacting than is the work of the cultural college. In the one case the standard to be reached is determined within the college circle and is more or less indefinite; in the other case, it is fixed by an outside, independent authority and is rigidly determined by the demands of the profession. Naturally, then, in our technical schools there is an exceptionally fine spirit for work, for close application to details, for a close co-ordination of laboratory and lecture room work, for clean-cut, usable knowledge, and for thorough training.

Not many years ago the mining school that offered a course in the general principles of underground work, of timbering, and of shaft sinking, with some little chemistry, and a superficial knowledge of geology, met the needs of students who desired to become mining engineers, and satisfied the demands of the mining industry. Within the last few years, however, the demands of the industry have become so broad and so deep that the older course satisfies neither the needs of the prospective mining engineer nor the demands of the profession.

Mining is now subdivided into coal mining, mining geology, non-metallic mining, metal mining, and metallurgy. The mining school of to-day which would be up to date must recognize all of these branches of mining. It must not make the mistake of allowing its graduates to be such one-sided specialists that, on leaving college, they can do only one thing. The function of the technical school is not to graduate unfledged specialists, but to develop a young man so that he can enter the profession at the bottom and eventually become a man of power and of resources. To this end a thorough grounding in fundamentals is absolutely necessary.

Long observation of men, trained and untrained, leads one to see that the mere practical man soon reaches the limit of his development. This is due primarily to the fact that his observations are limited, he cannot assimilate the condensed experience of others, and he lacks fundamental concepts and gen-

eral laws. He is filled merely with facts and personal experiences. The trained man, however, has the basal laws of physics, of chemistry, and of mathematics thoroughly implanted in him as a mental substructure into which all future facts and personal experience can be logically placed. In the one case there is, almost universally, a limit of development beyond which a man cannot go; in the other case, there need be no limit to growth and development. If this were not so, our technical schools would have no good reason for existence.

The mining school of to-day must not only feel the pulse of the mining industry but must recognize early the range of pulsation. It is not sufficient to be satisfied with the accomplishments of a decade ago. Coal mining, for example, has become a settled industry. The rewards are not great but they are fairly sure. The most recent demands in this branch of mining are for a thorough knowledge of the action of mine gases, the explosibility of coal dust, rescue work, and "first aid to the injured." A school giving a course in coal mining to-day would be sadly derelict in its duty if it did not include these subjects in its curriculum. The ideals for this work have risen steadily in the past decade.

The mining of the non-metallic minerals has been developing so rapidly of recent years that this branch of mining is even now demanding attention. The clays, shales, granites, lava stones, and marble are now being sought as never before. The cement industry is growing prodigiously and the uses of cement are becoming so numerous that only a specialist can become familiar with them.

Mining geology, also, has only recently come into its own. Herein lies the union of the mining engineer and the geologist. The old time prospector, with his burro, his outfit, and his "nose for ore," is gone forever. In his place is the trained mining geologist whose prospecting is done quite as much underground as on the surface, who understands the principles of ore deposition and the play of dynamic forces, and who can read earth changes with little to guide him. The mining industry needs this man and the schools should supply the material from which he can be developed.

But after all is said the field of ore dressing and metallurgy contains the great problems for the mining engineer. When coal has reached the surface it is ready for market, but when the metal miner brings his ore to the surface his troubles have only begun. If his ore can be shipped directly to the smelter he is merely relieved of the burden; he passes the problem on to the metallurgist. If his ore must be concentrated, or subjected to some form of local treatment like cyanidation, a new field of work opens before him. In this realm of ore treatment lie the serious problems of the future. Herein lies the great work of the mining schools—the work of training the future mill-man and the metallurgist.

The industry does not need a man versed merely in the theory of inorganic chemistry; there are many such who are not metallurgists. Certainly the metallurgist must be a master of applied inorganic chemistry, but he must be much more. Instead of thinking in grams he must think in tons; he must know not merely the refinements of the laboratory, but also the difficulties to be met in commercial practice. The steady decline in the cost of smelting during the past half century is a silent, though potent, witness to the improvements made. These have not been made by the practical man but by the trained man. There are many problems yet to be solved; it is the duty of the schools to develop the men who can solve these problems. Herein lies the ideal for the mining school. It is well known that neither the practical man working alone nor the theorist working alone can solve these problems. The man who has just the right combination is the ideal man.

The problem, then, before the schools is to make that combination of theory and practice which will develop the man to conquer the problems of ore treatment. Theoretical instruction combined with the ordinary laboratory is the combination generally used. The efficacy of this plan is doubtful. Some-

thing new is demanded in this age. That something is laboratory work on a full commercial scale, so that both the refinements of the laboratory and the difficulties of standard size work may be met and appreciated.

At the Colorado School of Mines this ideal is being realized by the erection and equipment of an experimental ore dressing and metallurgical plant, in which the equipment will all be of standard size and where ore in carload lots can be treated. In planning the building, the idea of flexibility has been kept constantly in mind so that any line of experimental work can be undertaken. The different sections cover ore dressing, cyanidation, roasting, and a section for metallurgical work. The plant will serve various purposes—a laboratory for the students; a place of research for advanced students, for members of the faculty, and for the mining engineer; and an experimental plant for the use of the mining industry.

The ideas involved in the plan are high but no higher than the needs of the industry demand. Since the great problems of the industry lie in the treatment of the ore, it is proper that the means adopted should include a great experimental plant for ore treatment. Such a plant will also provide adequate means for training the future metallurgist so that the school will be doing its duty not only to the mining industry but also to the young men who are looking forward to becoming the metallurgical engineers of the future.

*Colorado School of Mines.*

### The Manufacture of Standard and Alkali-Resisting Portland Cement in Colorado.

#### Editorial Correspondence.

The statistics of the United States Geological Survey show that the production of Portland cement in the United States made a new high record in 1910, notwithstanding the fact that prices were not satisfactory, particularly in the East. In 1900 the manufacture of Portland cement in the United States



FIG. 1.—NIOBRARA LIMESTONE BED AT PORTLAND, COLO.

amounted to 8,482,020 bbl. The production increased steadily and rapidly until, in 1909, the output was 63,508,471 bbl. This output was further increased in 1910 about 15 per cent to 20 per cent, and is estimated at 73,500,000 bbl. to 75,000,000 bbl.

Portland cement manufacture in Colorado began about twelve or thirteen years ago, and there was but one plant in operation until 1908, when a second was started. In 1909 the third commenced production of a special alkali-proof cement, and the prospects are that all three will be operated at capacity this year. The raw materials used for standard Portland cement in this State are a fairly pure limestone from the Niobrara formation (Fig. 1) and an overlying argillaceous cement rock locally called shale (Fig. 2), and so designated in these notes. These materials are exposed on either side of the Arkansas River between Pueblo and Cañon City, and two of the Colorado plants, those of the Colorado Portland Cement Company

and the United States Portland Cement Company, are located on the Arkansas River in Fremont County, at Portland and Concrete respectively.

#### The Portland Plant.

The Colorado Portland Cement Company operates a 3000-bbl. plant at Portland, Col., employing about 240 men under the direction of Mr. F. W. Brown, superintendent, and Mr. L. E. Foster, chemist. This plant is the oldest in the State. The accompanying illustrations show the lime and shale beds from which the raw materials are taken. Between the strata of limestone occur very thin seams of clay, which, in quarrying, break free from the limestone. It is, however, mixed with



FIG. 2.—SHALE BANK AT PORTLAND, COLO.

the spalled lime rock and sometimes used as part of the raw mixture.

Electric haulage is used for transportation to the stock house, where separate large bins are provided for lime rock, shale and spalls. These are drawn off through gates as needed and hauled to the crusher-room, where they are reduced by Gates No. 6 gyratories and rolls to about 1-in. size and elevated

to separate bins. From these the crushed materials are properly proportioned and diverted to blending bins located in another building. This series of 20 blending bins is a special feature peculiar to this plant, and provides a means for securing uniformity in the raw mixture, which is drawn from several bins at once. Thus any error which might have been made in proportioning the materials in one bin does not materially affect the composition of the raw mixture drawn from several.

The method of proportioning the raw cement mixture is as follows: Beneath the lime and shale bins are weighing hoppers, one for each ingredient. The shale carries about 67-68 per cent  $\text{CaCO}_3$  and the lime rock about 90 per cent. The shale is nearly a cement mixture, and lime rock is added in suitable proportion to make a mixture containing 74 per cent  $\text{CaCO}_3$ . The two materials are mixed in 8000-lb. lots, equivalent to about 12 bbl. of cement, and may contain from 7200 lb. to 7800 lb. of shale and 800 lb. to 200 lb. lime rock; or other proportions may be used according to the varying composition of the raw materials. A variation of 400 lb. of lime rock makes a change of about 1 per cent  $\text{CaCO}_3$  in the mixture.

By this system of controlling the mixture the effect of a change in composition can be determined in one to two hours according to running conditions in the plant; if both dryers are running the change will be effective in about an hour.

This practice of mixing the raw materials before comminution differs from that at the other Colorado plant, but it is apparently effective, inasmuch as the mixing continues through the dryers and conveyors. The dryers are coal-fired and are of the direct-heat rotary-blast type, 50 ft. long and 54



in. in diameter. The moisture is reduced to about 0.5 per cent.

The dried mixture is elevated to bins above the comminuters for preliminary grinding. A variety of grinding machines have been used at this plant, both on the raw and finishing sides, and there has been good opportunity to compare their work on different materials. For comminuters the preference at this plant favors the Jeffrey for raw mixtures and the Smith for clinker; as fine grinders, the Fuller mill is preferred for raw mixture and the tube mill for clinker. There is little preference between the Fuller and the tube mill for fine grinding of coal.

One Jeffrey and three Smith mills are in use as raw comminuters. Of their product 85 per cent passes a 20-mesh screen. This material is conveyed to bins above a battery of five Fuller mills and five tubes, all used for finishing the raw mixture. The Fuller mill has a capacity of about 15 bbl. of rock per hour and requires about 75 hp. The tube will grind about 22 bbl., with a power requirement of 125 hp. Colorado pebbles are used in the tubes, the consumption being at the rate of 45 tons per month, or 1 lb. per barrel. Screen tests of the finished raw mixture show 92 per cent through 100 mesh and 82 per cent through 200 mesh.

The kiln-room contains nine rotary kilns, six being 60 ft. and three 120 ft. long. The 60-ft. kilns are 6 ft. in diameter; one of the 120-ft. kilns is 9 ft. in diameter and the other two are 7 ft. at the feed end and 7.5 ft. at the discharge. The raw mixture enters the kiln with about 8 per cent added moisture and is burned at a temperature of about 2800° Fahr. Southern Colorado slack dried and ground to 90-95 per cent through 100-mesh screens is fired under 3-oz. pressure delivered by Sirocco fans. The coal consumption is 135-140 lb. per barrel of cement. The kilns are operated by variable-speed motors and require about 6 hp for the smaller and 35 hp for the larger.

The hot clinker elevated from the kiln-room may be diverted to the stock pile or, if needed immediately, is cooled in a rotary cooler, 6 ft. x 120 ft., operated by a 32-hp motor. Clinker can be withdrawn from the stock pile by conveyor operating in a tunnel between the clinker storage and the finishing end.

The finishing side of this plant parallels the raw side, the direction of flow of the material being reversed after the clinker is burned. On the finishing side three Smith comminuters and one No. 8 Krupp mill are used for preliminary grinding and eight tubes for finishing. The former grind about 85 per cent through 20-mesh. At this stage the retarder is added to control the setting time of the finished cement. Gypsum is used, and a sufficient quantity added to give about 1.6 per cent  $\text{SO}_3$  in the finished product.

The quantity of gypsum added is controlled by a ratchet-and-pawl device which operates a screw conveyor at the bottom of the gypsum bin. There are several pawls of different lengths, any one of which can be thrown into use, and by using a longer or shorter pawl the speed of the screw can be increased or diminished, with proportionate regulation of the quantity of gypsum discharged. The mixed cement and gypsum are elevated, weighed on a Richardson automatic scale, and conveyed by screws to hopper bins above the tube mills. The tubes grind about 94 per cent through a 100-mesh screen and 78-80 per cent through 200-mesh, which is well above the requirements established by engineering societies, viz., not less than 92 per cent through 100-mesh and 75 per cent through 200-mesh.

The chemical and physical tests made are those customary at cement plants. The raw mixture is sampled hourly at the various stages of its comminution, and tested for fineness and percentage of  $\text{CaCO}_3$ . An average day and night sample of the finished raw mixture is similarly tested. The finished cement is sampled hourly, analyzed for sulphur and tested for fineness, setting time, tensile strength and effect of boiling temperature. Carload shipments are controlled with tensile

strength tests on 7-day and 28-day briquets. The following figures are representative of the tensile strength of neat and sand (1:3) briquets, as developed in the times specified:

	24h.	7d.	28d.	90d.
Neat.....	345	735	850	850±
Sand.....	260	25 per cent gain	15 per cent gain	

The following analyses are typical at this plant:

	Raw Mix.	Clinker.	Finished Cement.
$\text{SiO}_2$ ..	13.6	23.0	22.3
$\text{Fe}_2\text{O}_3$ ..	5.8	3.2	3.2
$\text{Al}_2\text{O}_3$ ..	5.8	7.5	7.4
$\text{CaO}$ ..	42.3	64.5	64.0
$\text{MgO}$ ..	0.5	1.0	1.0
$\text{SO}_3$ ..	1.6	0.5	1.4
Ignition loss .....	36.8	0.1	0.1

The fuel used in the kilns carries about 18 per cent ash, and the determination of this factor is the only control exercised over coal purchases.

The company is making extensive improvements in its plant

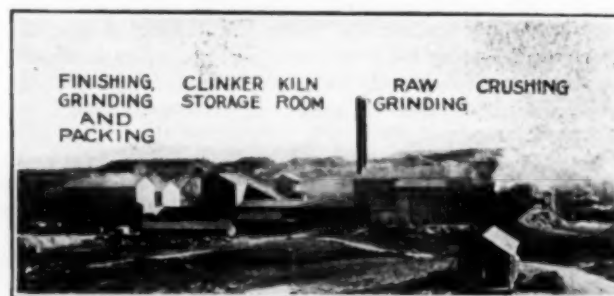


FIG. 3.—PLANT OF THE UNITED STATES PORTLAND CEMENT COMPANY, CONCRETE, COLO.

which will make for more economical operation. New crushing and drying equipment will be installed, and the power plant considerably enlarged, including the installation of a gas-producer plant. The cost of improvements will be \$200,000.

#### The Concrete Plant.

About one mile east of Portland is the town of Concrete and the cement plant of the United States Portland Cement Company (Fig. 3). This plant commenced production in 1908, almost a decade after the first operation of the Portland plant, and consequently is more modern in construction and in some features of operation. The plant has a capacity of 1500 bbl. per day and employs about 100 men. Operations are under the direction of Mr. William Fowden, superintendent, and Mr. I. J. Dilts, chemist.

The same raw materials are used as at Portland, with the exception that the lime is hand picked and the clay seams and spalls are rejected. The argillaceous material also is carefully selected, only that rich in lime carbonate being used. The clean limestone averages about 91.5 per cent  $\text{CaCO}_3$  and the clayey cement rock about 71.5 per cent. The plant is so situated with respect to the quarries that the rock is transported by gravity, and the empty cars returned by gasoline locomotive. The successive units of the plant are arranged in a straight line, differing in this respect from the plant at Portland, where the raw and finishing sides are parallel.

The rock is dumped directly to two Symons gyratories, Nos. 5 and 6. These crushers are the same as used in metallurgical plants in the West, a brief reference to which was made in our issue for February, 1911, page 92. The crushed rock is elevated to conical bottom steel bins, three of which are for shale and one for lime rock. These bins hold sufficient material for a 48-hour run of the plant. Each class of material is separately dried and ground before mixing, and in this respect the practice differs from that at Portland, where the raw materials are mixed immediately after crushing and before drying. Belt



conveyers from the bottom of each bin run to two Ruggles-Coles dryers, with exhaust fans and Cyclone dust separators attached.

One of the cardinal points of cement manufacture is the fine grinding and intimate mixing of the raw materials. The tendency at this plant is toward finer grinding, both on raw and finished products. One Williams and two Jeffrey mills are used as raw comminuters, grinding about 88-90 per cent through 20-mesh screen and 45 per cent through 100-mesh. The ground materials are elevated and diverted to bins, of which there are four placed in line, three for shale and one for lime rock. It is from these bins that the raw mixture is made and controlled by a system which is very accurate.

A short screw conveyor draws the material from each bin. The three screws on the shale bins are operated by one variable-speed induction motor, and deliver to a large screw conveyor running at right angles to them. The screw on the lime bin is similarly operated by a variable-speed motor, and likewise delivers to a second screw running at right angles to it. These cross-conveyors deliver to a common point where the lime and shale are mixed. The speed of the motors and screws is controlled by the chemist through rheostats placed in the laboratory. The standard mixture desired is one containing 75.5 per cent  $\text{CaCO}_3$ , and is made up of about 25 per cent lime rock and 75 per cent shale.

By previous experiment the chemist has determined the percentages of lime rock and shale which will be delivered at various speeds of the screws, also the lime carbonate content of the mixture which result from various combinations of speed of both motors. These figures have been tabulated with reference to the position of the pointers on the controlling rheostats, which thus become an arbitrary index of the mixture being made. The tabulations have been made for a combination of the one lime bin with two shale bins, or with all three shale bins.

Determinations for  $\text{CaCO}_3$  in the raw mixture are made every 10 minutes, and should the result be much higher or lower than the average desired, the chemist can, by reference to his tables, immediately set his rheostats at a combination which will correct the error. It will be seen that this system exercises a very close control over the composition of the raw mixture, which is another important point in cement manufacture. The effect of a change can be determined in from three to five minutes, and no great quantity of material of wrong composition can pass on to the bins; and even this quantity will become more thoroughly mixed in its subsequent treatment and cannot exercise much influence on the entire mixture.

Three 6-ft. x 22.5-ft. Smidth tube mills are used for fine grinding. A screen test of the tube-mill product shows 95 per cent passing a 100-mesh screen, which has been standard practice at this plant for over a year. The tendency is in favor of still finer grinding, however, and the probability is that all will be ground through 100 mesh.

Two Allis-Chalmers rotary kilns, 8 ft. x 125 ft., are fired with powdered coal delivered by blast through two 6-in. feed pipes. The fuel consumption is about 110 lb. of coal per barrel of cement. The total consumption of coal for drying and clinkering is about 130 lb. per barrel. Southern Colorado slack is used, dried in a Ruggles-Coles dryer and ground in Williams and tube mills. The Williams mill grinds about 86 per cent through 20 mesh and 40 per cent through 100 mesh; the tube grinds about 94 per cent through 100 mesh.

The provision for clinker storage is one of the features of this plant, a large area with concrete walls and floor being reserved for this purpose. The clinker is elevated from the kilns and distributed over the storage yard by a link-belt bucket conveyor running on a 100-ft. truss beam supported by steel towers. Beneath the floor are three lateral tunnels connecting with a main tunnel leading to the finishing unit. Hoppers are placed in the floor at suitable intervals along the lines

of the tunnels, and through these the clinker is drawn into scale cars in which it is weighed and trammed to the finishing department.

A weighed quantity of gypsum is then added from a scale hopper and the weight checked on the second beam of the tram car. The whole weight is then checked on platform scales. The quantity of gypsum added is variable, but amounts to about 12.5 lb. per barrel of cement, which gives about 1.5 per cent  $\text{SO}_3$  in the finished product.

In the finishing end Smidth comminuters and tube mills are used, there being two of each. The comminuters grind 88 per cent through 20 mesh and 40 per cent through 100 mesh; the tubes give a finished product of 98 per cent through 100-mesh and 82 per cent through 200-mesh screens. One of the advantages of clinker storage is noticed in the increased capacity of the machines used for fine grinding. When grinding weathered clinker the tubes will handle up to 20 per cent more material than when fresh clinker is used.

For the storage of finished cement four stock bins of 1250 barrels capacity each are placed in the packing house. Each bin is controlled with the usual tests before being released for packing and each car shipment is similarly tested. The usual routine test is applied at various stages in the process. The cement has an initial set in about 2.5 hours and final set in 5 hours. The boiling test is applied for 24 hours and the pats show no deterioration. A sulphur determination is made on the finished cement twice daily. The following figures show average tensile strength tests:

	24 hr.	7d.	28d.	90d.
Neat .....	350	650	800	850±
Sand (1:3) .....	290	325	375	

The finished cement analyzes in per cents:  $\text{SiO}_2$ , 22.75;  $\text{Fe}_2\text{O}_3$ , 2.25;  $\text{Al}_2\text{O}_3$ , 8.0;  $\text{CaO}$ , 64.0;  $\text{MgO}$ , 0.75 to 1.1;  $\text{SO}_3$ , 1.6.

The plant is operated by electric power generated at Cañon City 15 miles further west, but the company has under consideration the installation of a gas-producer plant.

#### Alkali-Proof Cement.

The manufacture of cement which will resist the corroding and disintegrating action of alkali soils and waters, and at the

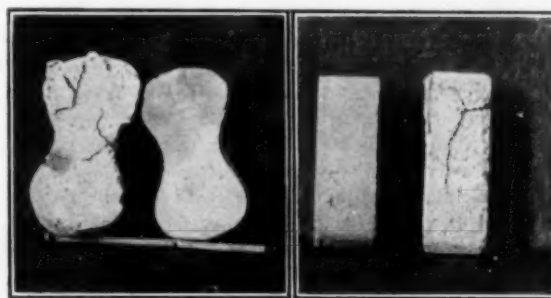


FIG. 4.—EFFECT OF STRONG ALKALI WATER ON STANDARD PORTLAND CEMENT BRIQUETS (TO THE LEFT) AND ALKALI-PROOF PORTLAND CEMENT (TO THE RIGHT).

same time develop the strength of standard Portland cement, is a new departure. The third Colorado cement plant mentioned before is that of the Western Cement Products Company, located at Minnequa, Pueblo, Col. The product is an alkali-proof cement, marketed by the Colorado Portland Cement Company, Denver.

Reference to the soil survey maps of the United States government will show the wide distribution of surface and sub-soil alkali in the western part of the country. Comparatively small areas of the Rocky Mountain region and adjacent plains are free from it, and inasmuch as this region includes some of the vast reclamation projects in which cement and concrete are extensively used the importance of an alkali-resisting cement becomes apparent.

The alkali usually present in Western soils has its origin in disintegrating rocks and in deposits of gypsum, and is mainly calcium sulphate. There are also present magnesium sulphate, and the sulphate, chloride and carbonate of sodium. The action of these salts on structures of ordinary Portland cement is disastrous, causing either partial or total destruction in a comparatively short time. This action may be shown in a laboratory way by making briquets of standard Portland cement and subjecting them to the action of strong water. In Fig. 4 is shown a comparison between briquets of standard Portland and the alkali-proof cement treated in this manner for 60 days. At the end of that time the standard cement briquet is shown to be cracked and on the verge of disintegration, while the briquet of the alkali-resisting cement is sound and strong.

The cause of this action of alkali solutions is found in the composition of standard Portland cement. It contains approximately three times as much lime as silica, iron and alumina, giving an excess of lime over the quantity required for chemical combination with the silica. This excess lime is attacked by the acid radicals of the alkali, forming compounds which, in their subsequent crystallization, expand and disintegrate the mass. Once begun this action continues until, by the processes of solution and mechanical erosion, the structure becomes honeycombed with cracks and pores and finally fails entirely. A peculiar feature of this disintegration is that it may not be apparent on the surface of the work until the central portion has failed. This may be due to the fact that the excess of lime in the surface has early combined with carbon dioxide, forming carbonate which is not attacked by the alkali. Capillary attraction, however, forces the alkali solution into the center, where it accomplishes its work of destruction.

The peculiar qualities of alkali-proof cement which give it resistance to alkali are the following: It contains less than twice as much lime as silica, all of the former being chemically combined and the latter in slight excess. Thus there is no free lime to be attacked by the alkali sulphates. The alkali-proof cement is finer than ordinary Portland, giving maximum strength and minimum permeability, with little or no opportunity for disintegrating solutions to enter the mass. It has a lower specific gravity (2.96) than standard Portland cement (3.1), and hence greater volume for a given weight.

In the physical tests alkali-proof cement shows a slower setting time and develops tensile strength more slowly, but more steadily and for a longer time than standard Portland. A comparison of the two will show about equal strengths at 7 to 10 days, after which the alkali-proof cement will increase more rapidly. The following tabular statement will illustrate this and also show that the new cement is ground much finer than standard products:

	Portland.	Alkali-Proof.		
Fineness.				
Per cent through 100 mesh..	94.9	98.8	98.6	99.1
Per cent through 200 mesh..	77.3	88.6	87.0	88.7
Setting Time.				
Initial .....	3:22	6:00	5:40	5:40
Final .....	7:13	7:45	7:55	8:20
Specific Gravity .....	3.13	2.96	...	...
Tensile Strength.				

Neat.				
24 hr. ....	408	220	225	170
7 d. ....	703	600	490	515
28 d. ....	763	975	865	860
Sand (1:3).				
7 d. ....	272	285	340	315
28 d. ....	361	445	540	475

Another interesting manner of testing the alkali-resisting properties of the new cement is to make briquets with sand and varying quantities of gypsum and test their strength from

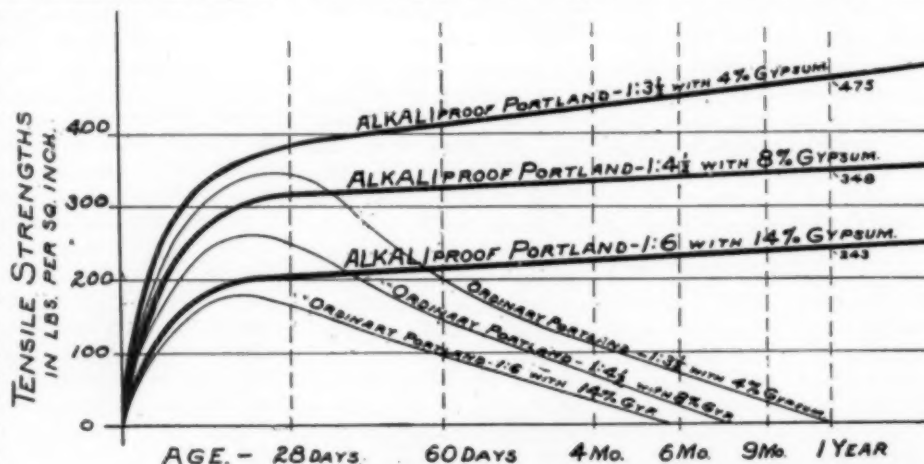


FIG. 5.—EFFECT OF ADDITION OF GYPSUM ON STRENGTH OF STANDARD AND ALKALI-PROOF PORTLAND CEMENT.

time to time. It will be found that ordinary cement will develop strength as usual for a short time, depending on conditions, and then suddenly deteriorate and ultimately disintegrate. On the other hand, the alkali-resisting cement will show a steady increase in strength over a long period. These points are illustrated in the accompanying curves, Fig. 5, which will require no further explanation.

Additional comparative data are obtained by exposing briquettes to the action of air, fresh and alkali water for various periods. The alkali solution used in the tests tabulated below is about five times the strength of naturally occurring alkali waters and has the following composition:

	Grains per Gal.
MgSO <sub>4</sub> .....	650
CaSO <sub>4</sub> .....	900
Na <sub>2</sub> CO <sub>3</sub> .....	200
NaCl .....	200
Na <sub>2</sub> SO <sub>4</sub> .....	1300

In testing briquets with this solution it is renewed fortnightly to keep up the strength. Below are the results of tests of four different cements exposed to air, fresh water and alkali solution. The abbreviation "dis" indicates disintegration.

	Tensile Strength Tests.			
	American Portland.	German Portland.	Special German.	Alkali-Proof.
7 days.				
1:3 air .....	310	142	112	335
water .....	267	165	145	342
alkali .....	175	185	160	342
1:5 air .....	158	120	90	212
water .....	160	122	92	195
alkali .....	128	93	140	208
28 days.				
1:3 air .....	270	197	205	320
water .....	280	165	247	390



alkali .....	342	180	240	405
1:5 air .....	242	175	185	190
water .....	140	140	175	245
alkali .....	dis	137	138	235
90 days.				
1:3 air .....	455	268	278	367
water .....	405	320	237	452
alkali .....	dis	370	310	505
1:5 air .....	237	165	145	205
water .....	215	168	170	240
alkali .....	dis	175	180	245
6 months.				
1:3 air .....	465	335	270	440
water .....	415	335	245	490
alkali .....	dis	dis	215	480
1:5 air .....	255	175	155	205
water .....	225	180	175	245
alkali .....	dis	dis	dis	250

It will be observed that each of the ordinary cements disintegrated at or after 28 days, while the other remained sound. Another peculiar fact noticed from a study of several thousand tests is that just prior to disintegration the standard Portlands show a temporary increase in tensile strength.

#### Process of Manufacture.

Alkali-proof cement is a mechanical mixture of cement materials, and in this one respect resembles pozzuolan cement. Both cements contain low lime and high silica, but the pozzuolan develops comparatively low strengths while the other is equal or even superior to ordinary Portland in this respect, with the additional value of resisting the attack of alkali. The ingredients used in the alkali-proof mixture are iron blast-furnace slag, standard Portland cement and a mixture, the exact nature of which is not given, but which contains gypsum and other materials. The use of this mixture gives the slag increased hydraulic value and makes the cement strong as well as resistant to alkali.

The mixing of these ingredients is controlled so as to give a cement which will pass standard specifications as to strength, while giving maximum resistance to alkali. The excess lime in the ordinary Portland and the soluble silica in the slag are determined chemically and enough slag is used to provide an excess of silica over the quantity required to hold all of the lime in combination. The ingredients are then intimately mixed and finely ground. The resulting cement does not deteriorate with storage and finds special use in the field mentioned.

The plant at Minnequa has a capacity of 500 barrels daily and is under the direction of a technical staff including Tracy Bartholomew, manager; O. P. Harpel, superintendent, and H. C. Emery, chemist. Slag is obtained from the furnaces of the Colorado Fuel & Iron Company, being granulated at the furnace and dried, screened and ground to about 30 mesh at the cement plant. After being properly mixed the ingredients are ground in a tube mill with Colorado pebbles and "cylpebs" in different compartments of the same tube. The product has the following composition:  $\text{SiO}_2$ , 32.9;  $\text{Fe}_2\text{O}_3$  and  $\text{Al}_2\text{O}_3$ , 13.2;  $\text{CaO}$ , 48.0;  $\text{MgO}$ , 2.5;  $\text{SO}_3$ , 1.8; and ignition loss, 0.3 per cent. It is the result of about seven years' study and experiment, during which time it has been subjected to many practical tests.

In conclusion, acknowledgment is due the members of the staff at each of the plants for their courtesy in giving the data presented herewith.

Inside amalgamation of free gold is not practised except under special conditions which seem to warrant its use. In case the outside plates become covered with a slimy film which cannot be easily removed, it may be profitable to apply inside amalgamation, with the frequent addition of mercury.

#### Manufacture of Carbon Electrodes.

In the London *Electrical Review* of Jan. 20 Mr. WILLIAM CLACHER, F. C. S., gives some details of the manufacture of carbon electrodes as carried out at the works of the British Aluminium Company, Ltd., in Greenock and Kinlochleven.

The author first gives the general methods used at Greenock. At Kinlochleven the methods have been somewhat modified, due to improvement in the plant, but the principles underlying both systems are the same. The difference in favor of the carbon produced at Kinlochleven is due to the new furnace there erected, which is of the regenerative type, giving a much higher temperature than the Greenock tunnel furnace.

The properties essential to a good carbon are: High conductivity, non-liability to crack, efficient contact arrangement between the carbon and the metal conductor.

The characteristics accompanying high conductivity are: (a) a good ringing sound when struck; (b) a high specific gravity and low porosity. The specific gravity of different forms of carbons varies thus:

Calcined anthracite has a specific gravity of.....1.80

Calcined petroleum coke has a specific gravity of....2.00

Kilned retort carbon has a specific gravity of.....2.05

Taking blocks made from these carbons with the same porosity, the retort-carbon block has the highest conductivity; the petroleum-carbon block has the next highest conductivity, and the anthracite-carbon block the lowest. This refers to anthracite calcined at 1000° C., not at 3000° C.—that is, non-graphitized anthracite.

The outline of the process for making the carbon is as follows:

1. The petroleum coke is first calcined to remove volatile matter and increase the specific gravity and conductivity.
2. The calcined material is pulverized.
3. The pulverized material is mixed with the binding medium in a steam-jacketed mixer.
4. The mixed mass is molded.
5. The block is kilned in the furnace, the kilning taking five days. After this the blocks are cooled, brushed, and stocked ready for use.

The petroleum coke is obtained in the distillation of shale oil; it is the residue left in the iron retorts. There is left in it from 5 per cent to 13 per cent of volatile hydrocarbons, which are not driven off with the heavy oils. There is also a small amount of inorganic material averaging 2 per cent. Most of the latter is in the harder parts; the softer and more porous parts are almost ashless. Mr. Clacher has analyzed pieces with a difference of over 5 per cent between the separate portions. Surface coloring he has found to indicate the presence of inorganic material; such carbons are none the less considered of lower grade.

The pieces as they come from the oil works are usually of large size. This is good for stacking purposes. There is at the same time a percentage of smalls and dust, varying with the quality of the coke, the softer coke having by far the largest amount. This is a serious matter to the Aluminium company; as the small dust chokes the calciners and is not easily worked away, only a certain amount can be added with each charge of coke.

The calciners are vertical rectangular brickwork chimney-like constructions; the coke is put in at the top and the calcined material drawn out with long rakes at the bottom. The charges are given every two hours, and a half charge is drawn every hour; 2 cwt. (101.6 kg or 224 lb.) was the charge in the Greenock calciners. The time the coke takes to travel from the top to the bottom is five hours.

The temperature in the calciner is approximately 2000° C. at its hottest part. The calcining is a very costly and, as at present carried out, wasteful process, for 30 per cent of the coke put in is burned away during the process; the volatile

matter in the coke before calcining averages 8 per cent, so that 22 per cent of carbon is used to raise the temperature to 2000° C., representing fuel at \$17.50 per ton—considerably more costly than the best smokeless coal.

Calcination has been carried out in horizontal retorts; these are more expensive at first, but they are much more economical with the coke, 20 per cent being the loss usually found with them. In cases where the calcination was effected in a current of producer gas the loss was little more than the percentage of volatile matters in the material; 15 per cent was the figure obtained from 10 per cent volatile in the coke, and the temperature is nothing like as high as in the vertical retorts, not exceeding 1000° C. The reports on anodes made from coke calcined thus were of a satisfactory character; the specific gravity of the carbon was slightly less than that calcined in the vertical calciners, 1.95 being obtained.

Calcined coke was also once made by packing the blocks before entering the tunnel furnace with it, and then passing it through; the coke swelled up and the blocks thus packed were unusable.

In the vertical calciners experiments were begun before Mr. Clacher left with the object of saving the excessive loss by passing producer gas into the calciners and using that in place of the coke to get the required temperature. The work was not far advanced when he left, and he is not aware that anything on these lines has been worked at by the Kinlochleven staff.

After calcination the coke at Greenock contained 0.7 per cent of inorganic material, consisting of 0.5 per cent silica, 0.15 per cent oxides of iron and alumina, and 0.05 per cent soluble sodium salts.

At Kinlochleven the arrangements of the stores for the oil coke and the transporting and grinding machinery have been totally altered. In addition, breakers to crush the coke to the size required for calcining have been erected; vertical rotary crushers and high-speed vertical grinding mills are used, and the sieving machinery is more up to date. At Greenock the breaking of the coke was done by men with hammers.

The grinding of the calcined oil coke is done in two stages: (1) An initial crushing to pass through a  $\frac{3}{8}$ -in. mesh; (2) a grinding to get the correct gristing of the powder. The gristing is an exceedingly important item; the amount of fine dust and large must be properly proportioned to get the block with the highest specific gravity and least porosity.

A good gristing is: Through 100-mesh, 40; 60-mesh, 15; 30-mesh, 20; 16-mesh, 15; between 16-mesh and 8-mesh, 10.

The cost of grinding at Greenock was from 75 cents to \$1 per ton; there was a large loss of carbon in the open pan-mill and the machine for the finer grinding amounting to 7 per cent of the total. This loss is done away with when using the new machinery.

The supply of petroleum coke, a by-product in the paraffin industry, is dependent on the quantity of shale oil distilled, and this is strictly limited in amount.

In view of an expected increase in the demand for aluminium and a consequent greater need for a good carbon, a search was made for likely sources. Of retort carbon, which is used for making the furnace linings, the supply is limited to the quantity of coal gas made. Of anthracite there is an unlimited store, and provided the inorganic contents were within the limits of the supply of calcined oil coke, this was considered a possibly valuable article.

The limits of impurity allowed were 0.5 per cent silica and 0.2 per cent iron oxide; this was the average analysis of the ash contents and the kilned blocks at the time, in the year 1904.

A great amount of work was performed on this for a period extending over three years. After that the work was stopped for a time. After Mr. Clacher had left the company he continued the experimental work, and after overcoming some of the difficulties left unsolved in the Greenock factory, he patented the process (British patent No. 2071, 1909).

The following forms of carbon were treated at Greenock:

1. The returned blocks from the Foyers furnaces.
2. Retort carbon.
3. Anthracite, Scotch and Welsh.
4. Bituminous coal.
5. Graphite.

The work on (5) was carried on exclusively after Mr. Clacher had left the company.

1. The returned blocks from Foyers contained approximately 2 per cent inorganic constituents, of which a little over 1 per cent was silica and iron oxide; this was crushed to pass through an 8-mesh sieve, with the usual gristing and, after treating, the inorganic contents were reduced to 0.5 per cent. An analysis of the untreated and treated material is given as follows:

	Before.	After.
Ash .....	1.56%	0.53%
Soluble .....	0.33	0.17
Silica .....	0.46	0.14
Iron oxide .....	0.26	0.14
Oxide of aluminium.....	0.39	...
Difference (CaO, MgO, SO <sub>3</sub> )....	0.12	...

2. Gas retort carbon as used at Greenock contained from 2 per cent to 4½ per cent of inorganic constituents. Crushed to pass through an eight mesh, and of the usual grist, the inorganic was reduced from 3 per cent to 0.75 per cent. Analyses:

	Before.	After.
Ash .....	2.35%	0.68%
Soluble .....	0.10	...
Silica .....	0.91	0.20
Iron oxide .....	0.62	0.26
Oxide of aluminium.....	0.41	0.08
Lime .....	0.17	...
Magnesia .....	nil	...
Sulphur trioxide .....	0.11	...
Difference .....	0.03	...

#### Another analysis:

	Before.	After.
Ash .....	2.87%	0.71%
Soluble .....	0.08	...
Silica .....	1.13	0.16
Iron oxide .....	0.80	0.19
Oxide of aluminium.....	0.60	0.36 <sup>1</sup>

3. Scotch Anthracite.—The seams tested contained from 4 per cent to 12 per cent inorganic. Analyses are given of some purified; in this case of somewhat smaller grist, the whole passing a 16 sieve:

	Before.	After.
Ash .....	9.24%	1.81%
Silica .....	5.00	0.83
Iron oxide .....	1.06	0.22
Oxide of aluminium.....	2.02	...
Lime, magnesia sulphate and soluble .....	1.16	0.19
	Before.	After.
Ash .....	4.99%	1.08%
Silica .....	2.47	0.49
Iron oxide .....	0.46	0.08
Oxide of aluminium.....	1.69	0.19
Lime .....	0.09	...
Magnesia .....	0.05	...
Sulphur trioxide .....	0.08	0.32
Difference .....	0.15	...

Ground to pass through a 60 mesh the totals were reduced in proportion. Analyses:

	Before.	After.
Ash .....	6.50%	1.25%
Soluble .....	0.17	included in diff.
Silica .....	3.55	0.38
Oxide of iron.....	0.88	0.05
Oxide of aluminium.....	1.55	0.31
Difference .....	0.35	0.51

<sup>1</sup>Soluble and oxide of aluminium. z



With Welsh anthracite he got from a seam analyzed:

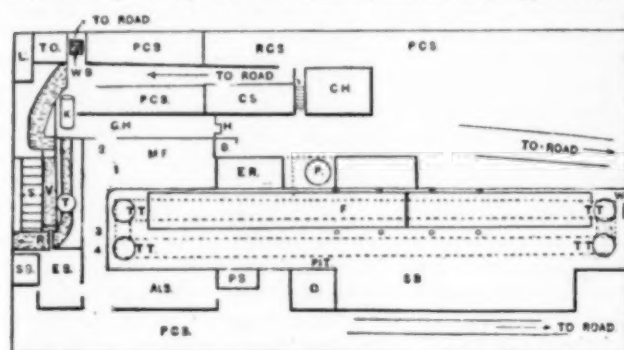
	Before.	After.
Ash .....	2.08%	0.57%
Silica .....	0.78	0.20
Oxide of iron.....	0.32	0.12
Oxide of aluminium.....	0.43	0.12
Soluble and difference.....	0.53	0.13

The material was milled to pass a 16 mesh.

4. With a good bituminous coal containing 3 per cent of ash the silica was reduced to less than 1 per cent. The iron oxide was not removed, and work was discontinued on that account.

The process used was a hot digestion with a strong caustic solution and the removal of the silicates, sparingly soluble in caustic, with a weak acid solution—iron oxide being subsequently extracted.

The accompanying diagram is a sketch plan of the general arrangement of the carbon factory; PCS is the petroleum coke stock; RCS retort carbon stock; CH, calcining house; GH, grinding house; H, hoist; MF, molding floor; F, tunnel furnace; TT, turntables; W, winch; SB, stock finished blocks; PS, stock pitch; CS, coal; B, boiler; ER, engine-room;



SKETCH PLAN, SHOWING GENERAL ARRANGEMENT OF CARBON FACTORY.

P, producer; L, laboratory; TO, time office; ES, engineers' shop; SS, smiths' shops; ALS, aluminium stock; O, offices; S, flight of stairs; V, overflow; R, runaway from turbine; T, water turbine; WB, weigh bridge; K, kilns for drying carbon. 1, 2, 3 and 4 represent the position of the machines, etc., shown in Figs. 1 to 4 respectively.

From the grinding mills the fine powder is taken away to the store hoppers, from which it is transferred to the Werner and Pfleiderer mixing machines; these are seen in Fig. 2 and are of the steam-jacketed kneading type, with reversing gear for changing the direction of motion. The temperature of the mixture when ready for discharging is 90° C. In the mixers shown the tipping is effected by means of an upright screw; in the large new mixers for Kinlochleven a worm quadrant and screw is used in place of the upright screw. The mixers in the illustration hold a charge of 300 lb. of mixture.

The hopper for weighing the carbon is seen over the right-hand mixer and the pitch-weighing tank over the left-hand mixer. The amount of pitch used is 22 per cent of the mixture; it is a medium hard brand and has volatile constituents to the extent of 50 per cent, as volatile is determined quickly; only 40 per cent volatile goes away during the slow kilning.

The tar used for making the linings for the electric furnace is much softer, containing up to 60 per cent volatile as determined by the quick method. This lining for the furnaces was made at the mixers in Greenock, bagged, and sent to Foyers, where it was heated and rammed in the furnaces at the side and bottom and then baked with current passed through before charging the furnaces with the electrolyte. The life of the linings varied from a few days to months. Latterly the mixture for the furnaces was made at the electric reduction works, saving the trouble and expense of the reheating.

For making the tar to the required degree Mr. Clacher used an instrument for measuring its fluid quality at the temperature of mixing (115° C.), the time a given weight took to run out of an orifice giving the indication required. He found tests so performed were usually in agreement with the volatile determination and a test of the fluid nature is of more value as a criterion for the binding material in the tar than is the percentage of volatile matter in the semi-liquid.

The bending point of a slab of pitch 3 in. long and ½ in. square is a very good test for the pitch. This test is certain;

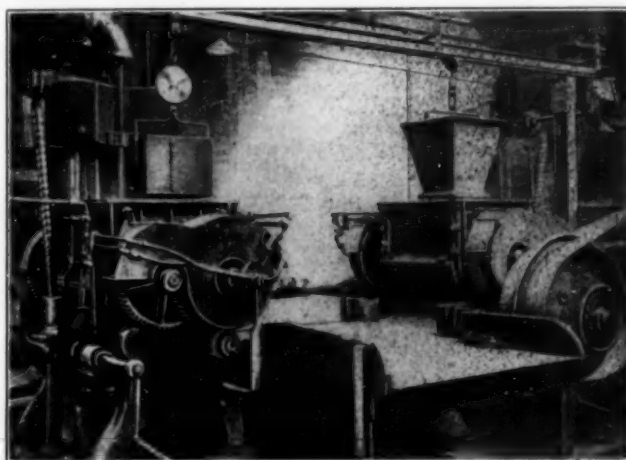


FIG. 1.—MIXING MACHINES.

the determination of the volatile content is anything but certain. Mr. Clacher has found that when the greatest care is taken the limit of accuracy of volatile determination is such that a determination of that kind is not a standard of value.

Fig. 2 shows the arrangement of the mixing and molding floor. On the right-hand side are two melted-pitch tanks; they are wrought-iron vessels, steam heated by internal coils. The pitch is lifted up by means of a small windlass and emptied into the tanks as required. A third tank was afterward added; this was a steam-jacketed vessel and worked more satisfac-

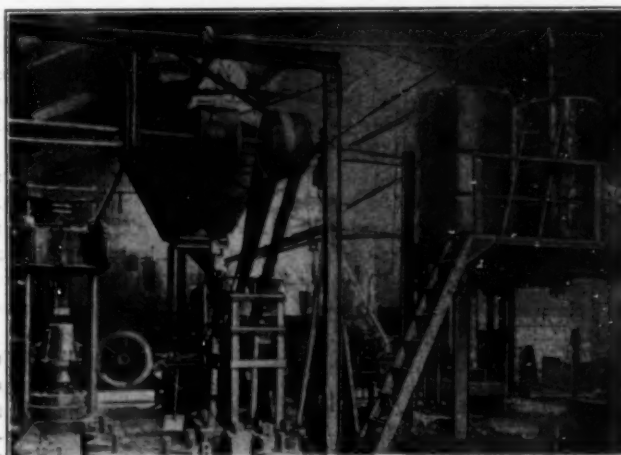


FIG. 2.—MIXERS AND PITCH TANKS.

torily than the two shown; the leakage of the joints in the coils caused frothing and overflowing of the tanks heated with the internal coils. In the center are the two mixers and the store hopper containing the carbon; on the left hand are the hydraulic molding machine and some of the molded blocks.

The mixture when tipped from the mixer to the platform is transferred in scoops to the container of the mold, which is in the center of the machine. When the mold is full the water valve is opened, the container rises and the block is formed.

When the gage registers 1 ton to the square inch the valve is shut and the runaway valve opened, the container lifted and the block discharged; it is taken away and put on the floor, there to remain for a day to harden previously to being packed in the trucks for putting through the furnace. Half a ton to the square inch will form a good block. One ton is usual, and as much as 4 tons has been used.

It is not possible to make a good block with mixture that is cool, and arrangements are made to work with as little stock of mixture as possible. The capacity of the machine illustrated was 40 blocks an hour. The Kinlochleven machine is more complicated and is designed for a larger output.

The blocks are 10 in. square, with cutaway corners, and are

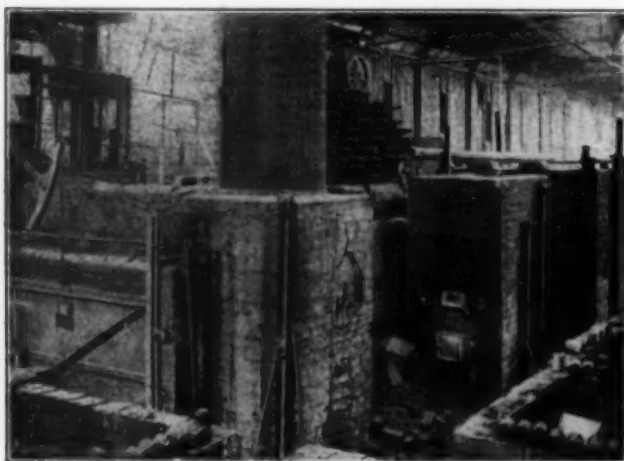


FIG. 3.—HEATING END OF FURNACE.

12 in. high. Those shown in the illustration are flat topped. This has been improved and for some time all the blocks have been made with the top rounded. The top of the block, efficient only as a means of making contact, is thus reduced in weight, and consequently a rounded-top block of equal weight to a square-top block has more carbon usable for the reduction of the alumina.

The contacts seen in the blocks are of mild steel and are molded in the blocks, the internal end being forked; the forks are thoroughly cleaned and coated with pitch previously to being put in the clamp for molding. It is noticeable that the contacts are not central in the block. When the contact is made at the furnaces the point of suspension becomes central and the blocks hang vertically without strain.

The form of contact shown is the best between the carbon and the metal; but these are very expensive to have in stock. A large number is required, and the wear and tear on them is great. The later forms of contact are made by screwing the metal into a molded thread in the block; the number of contacts required in stock is much lessened and the wear and tear comparatively inappreciable.

The weight of a block before kilning averages 70 lb. with the claw; after kilning, 66 lb. with the claw. The claw weighs 8½ lb. The loss of 4 lb. in the block corresponds to 8 per cent on the block and 40 per cent on the pitch used. The arrangement of the pitch tanks at Kinlochleven was much altered and in no way resembles the Greenock arrangement. This was on account of the much larger output that was expected.

When made the blocks are stored till they are hardened somewhat and are then placed in the trucks for calcining. At Greenock the blocks were stored on the molding floor; a few days' stock was kept.

The trucks seen in Figs. 3 and 4 are oblong and made of firebrick; they are mounted on bogies. The blocks are placed, 42 in number, in a truck and are packed with the fine ashes free from dust; the metal contacts are covered with fireclay covers and the fireclay covers are completely hidden with the

ashes. The truck is then pushed into the furnace and at the same time a truck with the kilned blocks is pushed out of the other end of the tunnel by the same movement, which moves each truck in the furnace one stage on. A hydraulic ram is used for this purpose. As soon as the truck with the unkilned blocks is in and the truck with the kilned blocks out the iron furnace doors are dropped and the furnace is at normal again.

Producer gas is the fuel used for heating the furnace, this being made in a Mason producer; the coal found best for that purpose was a Lanarkshire singles containing 3 per cent to 5 per cent of inorganic and 30 per cent volatile constituents; the analysis of the gas made is:

Carbon dioxide .....	4.0
Carbon monoxide .....	22.0
Hydrogen .....	16.0
Methane .....	4.0
Oxygen .....	2.0
Nitrogen .....	52.0
	100.0

The flues from the producer to the furnace require cleaning once every eight days; the regulation of the quality and quantity of the gas also requires attention. When Mr. Clacher went to Greenock the producer was consuming 40 tons of singles a week. By regulating the feed and the steam and the air he got the consumption down to 27 tons per week. At the same time a more constant temperature was produced.

The furnace is seen in Figs. 3 and 4. It is 200 ft. long. On either side are four gas valves. These are regulated to give an even temperature over the middle of the furnace. The escaping gases heat the incoming trucks and the outgoing truck is somewhat cooled before being ejected. The maximum temperature at the top of the truck is usually 1150°, at the bottom of the truck 700° C. The latter is the temperature to which the bottoms of the anodes made at Greenock are kilned. This is

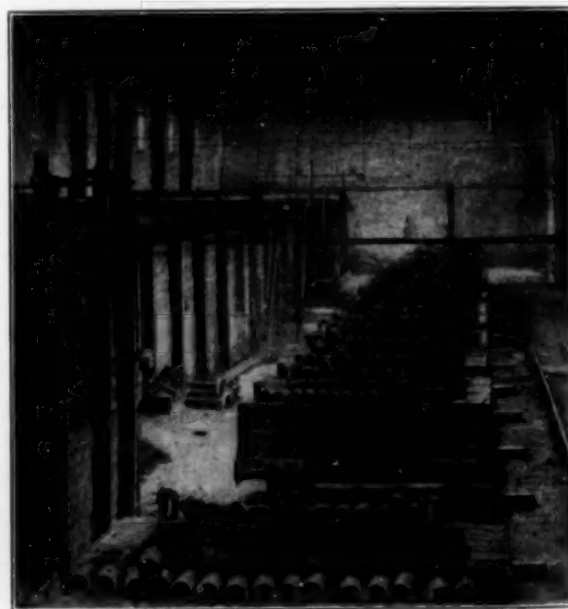


FIG. 4.—TUNNEL FURNACE AND ROW OF TRUCKS.

doubtless a disadvantage for the purpose to which the furnace is now applied—the calcination of alumina to be used at the electric furnaces.

The furnace at Kinlochleven is built of firebrick and is of the type used at Witton, in that it is regenerative, the chambers in which the articles kilned are cooling being used to heat the air for the combustion in the hot chambers; the temperature normally attained as a maximum is said to be 1400° C., and the



same time is taken for the cycle of operations as at the Greenock factory—5½ days. The chambers are filled with the boxes containing the blocks, which are packed as at Greenock with fine ashes; they are stacked one above the other in the chamber. The boxes are of fireclay and are made at Kinlochleven. When the chamber is filled to the requisite capacity the opening is sealed with brickwork, and this chamber is then made the last in the circuit of the gases passing through it previous to entering the chimney flues. The accompanying plan, Fig. 5, gives an idea of the arrangement of the chambers.

The blocks kilned in this furnace are said to be much more efficient at the electric furnaces than the Greenock kilned ones.

At Foyers during the past few years the Greenock blocks gave a yield of 1 lb. aluminium for 0.7 lb. carbon used; the most that is expected is 0.64 lb.

A great amount of trouble has sometimes been caused by the burning of the blocks by the air at the electric furnaces; this

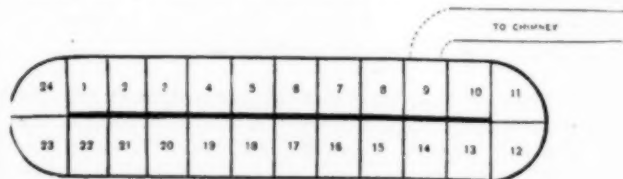


FIG. 5.—PLAN OF REGENERATIVE FURNACE.

serious reduction of the efficiency of the carbon was gone into, and many coatings were given the blocks to prevent the oxidation, whitewash among others. Mr. Clacher has reason to think this difficulty is solved. Cracking of the blocks at the electric furnace is a trouble due to the block not being to standard. A block should last from 100 hours to 140 hours, and when conditions are normal in the furnace the time factor is most valuable. When blocks last less than 100 hours they are not classed good.

The specific gravity of the Greenock blocks was 1.63 after kilning, and the porosity 20 per cent; different parts of the Greenock blocks had often different porosities and considering the difference in temperature between 1150° and 700°, the temperatures at the top and floor of the truck, this is not to be wondered at. Before kilning the specific gravity was 1.53.

Mr. Clacher experimented with the making of blocks from a very heavy pitch with less than 40 per cent volatile matter. Blocks thus made in a gas-heated mixer were harder than the usual block; their conductivity he did not have tested.

[In connection with this article it may be interesting to refer to some progress which is reported as having recently been made in the manufacture of large amorphous carbon electrodes. It is reported by reliable correspondents that in European electric steel furnace plants amorphous carbon electrodes in sizes up to 600 mm (24 in.) diameter are in use and that no troubles due to cracking of electrodes are experienced.]

The Arbuckle process of ore treatment as used in South Africa is thus described by H. A. White, in a paper read before the S. A. Association for the Advancement of Science: It involves the treatment of the pulp as a whole without separating sand from slime, and thus solves that difficulty by the method of evasion. The pulp, mixed with about three times its weight of weak KCN solution, is agitated by an air lift in deep tanks with conical bottoms. After about 20 hours' agitation, the whole pulp is transferred to a separating apparatus, where the solid portion, with about 25 per cent moisture, is drawn off for further washing, and the clear solution sent to the precipitation tanks. The drawback lies in the agitation of the whole pulp for the entire period, when perhaps the slime portion would not require so long an agitation. Nevertheless, it is possible that owing to decrease in working costs in larger units, greater skill in handling material, and other important reasons, this method, with modifications, may become of importance in South Africa.

## A New Form of Gauze Electrode for the Rapid Electrolytic Determination of Lead and Copper.

BY R. C. BENNER.

The present trend for lead and copper determinations being toward the use of electrolytic methods, any increase in rapidity, simplification or lightening of labor in the use of this method is of vital importance.

When the interfering elements are removed and the deposition is carefully made, the electrolytic determination of copper is generally recognized as the standard, and is, as well, the most accurate of the various methods employed in technical work. The determination of lead by this method until recently has been more likely to error than that of copper, as the  $PbO_2$  deposited was likely to have a varying composition depending greatly on the conditions of precipitation.

Every person who has made a quantitative, electrolytical determination of lead has found necessary the use of some factor other than the theoretical. This has made the determination of an arbitrary factor necessary in each case. When the peroxide of lead is deposited on the inside of a platinum dish it is possible to ignite the  $PbO_2$ , changing it to  $PbO$ , in which form it can be weighed with faultless results.<sup>1</sup>

When the deposition is made on the gauze electrode it is impossible to ignite over the Bunsen burner, as some of the  $PbO$  is volatilized, or some of the  $PbO_2$  remains undecomposed,

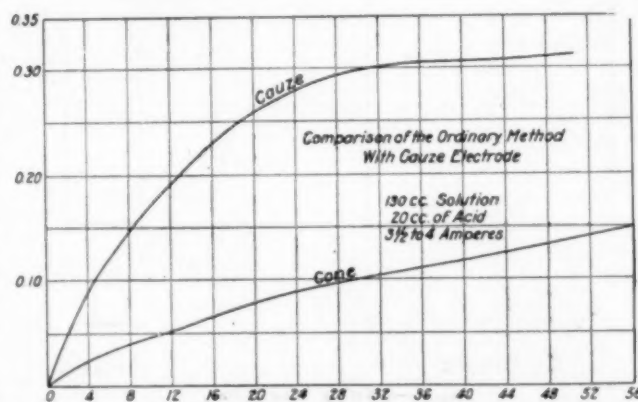


FIG. 1.—COMPARISON OF ORDINARY ELECTROANALYTIC METHOD WITH THE USE OF GAUZE ELECTRODE.

giving low or high results respectively. This may be obviated with the gauze electrode by igniting in a muffle below a red heat (500° C.) or in a small electric furnace made for the purpose.

Recently Sand<sup>2</sup> has been enabled to deposit the peroxide of lead in such a condition that the error caused by the use of the theoretical factor can be neglected when the amounts of lead found in a 0.5 gram sample of ore are taken. This he accomplished by means of a rotating cathode and gauze anode, depositing the lead from a strong nitric acid solution, free from  $NH_4NO_3$  and the oxides of nitrogen, at a temperature just below the boiling point (94-97° C.), using a current of 5 amperes. When the deposition is made in this manner the use of the factor 0.8651 gives correct results. This factor is so near the theoretical (0.866) that the latter can be used without error larger than other errors involved.

The simplest form of apparatus in use at the present time consists of the two ordinary platinum electrodes, with which it takes from five to ten hours to deposit the amount of copper found in 0.5 gram of ore. The shapes of these electrodes have been modified indefinitely but serve essentially the same purpose. The rapid precipitation of the various elements in such a form that they can be weighed accurately is possible with

<sup>1</sup>Treadwell Analytical Chemistry, New York, 1904, Vol. 2, p. 140.

<sup>2</sup>Chemical News, 100-269.

several different forms of apparatus, all of which are dependent on the necessity of agitating the electrolyte, so that as long as there is any metal in the solution some of the metallic ions will be touching the cathode at any instant and be deposited.

In order to accomplish this, Smith, at University of Pennsylvania, rotated the anode, while Gooch, at Yale, rotated the cathode. Both of these forms, however, while giving a rapid method for obtaining excellent results, possess the great dis-

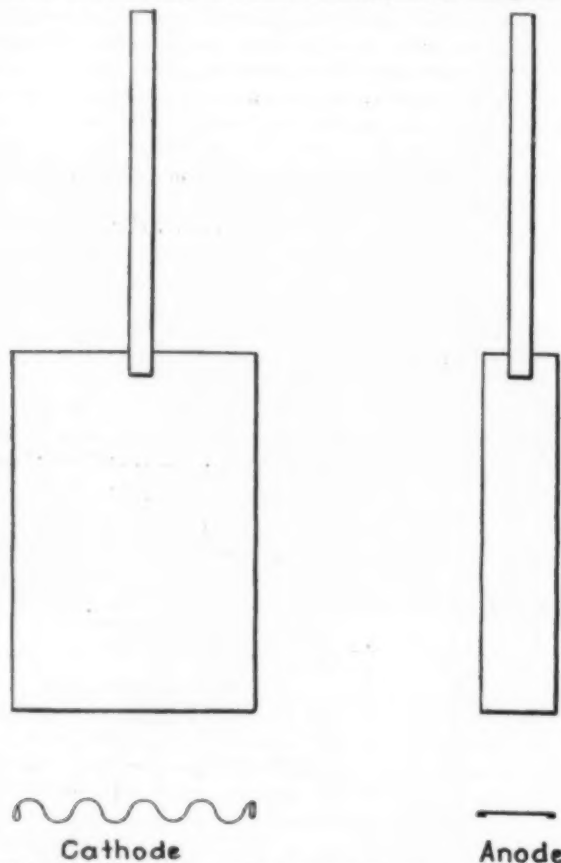


FIG. 2.—FORM OF ELECTRODES.

advantage of requiring more or less complicated pieces of apparatus with which to agitate the electrolyte, and more attention than the ordinary apparatus in use with the slower methods.

The advantage of rapidity for technical laboratories is more than counteracted by the disadvantage presented in this complexity. The need, then, of apparatus which combines the advantages of both and eliminates their disadvantages, becomes evident. It recently has been possible to accomplish this to a certain extent by the use of a cathode made of platinum wire gauze.<sup>1</sup>

The gauze electrode makes possible the use of stationary electrodes as simple as the most simple foil electrodes, at the same time allowing the deposition of copper and lead in a much shorter period than in the case of the ordinary electrodes. This will be seen by examination of the curves, Fig. 1.<sup>2</sup>

The deposition on the gauze electrode, even though it can be made much more quickly than by the old method, cannot equal the rapidity of the rotating cathode or anode. The nearest approach to the rapidity of the gauze electrode when stationary electrodes are used is found in the device of Messrs. Guess and Haultain. This consists in the use of very light electrodes, 1.2 grams in weight, and the precipitation of small

quantities of copper (less than 0.1?) in the presence of a nitro compound. Using a current of 1.5 amp it takes from 1.5 to 3 hours to make a deposition. The apparatus is very compact and handy for this work and has been put into use by many assayers through the western United States.

As this apparatus has such an extended use it seemed desirable to design an electrode which would be suited for use in connection with it and yet possess some of the advantages of the cylindrical gauze electrode employed for our previous work on lead and copper. The electrode designed and used in our experiments is shown in Fig. 2 and consists of sand-



FIG. 3a.—WIRING ARRANGEMENT.

blasted gauze. These electrodes can be connected by the method shown in Fig. 3, a, b, and c, which is used in the Guess-Haultain cabinets, or by the arrangement shown in Fig. 4, used by Dr. W. H. Ross and the author for general electrolytic work. By this arrangement it is possible to use each deposition cell independently. In Fig. 4,  $R_1$ ,  $R_2$ ,  $R_3$ , are small resistance boards of about 10 ohms, made of nichrome resis-

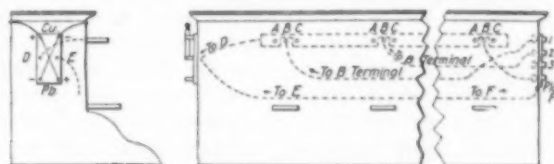


FIG. 3b.—WIRING ARRANGEMENT.

tance wire, manufactured by the Driver Harris Wire Co., of Harrison, N. J. This is the most satisfactory resistance wire with which the author has had experience. Each deposition cell has its resistance box for independent regulation. The amount of current for the system as a whole is regulated according to circumstances. The switches,  $S_1$ ,  $S_2$ ,  $S_3$ , are the main switches for opening and closing the circuit for the

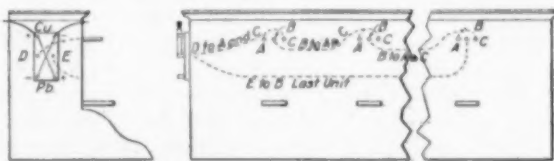


FIG. 3c.—WIRING ARRANGEMENT.

operation of each cell. When a cell ( $C_1$ ) is in operation alone, the other switches  $S_2$  and  $S_3$  remain open when  $S_1$  is closed.  $AS_1$  is the ammeter switch. By closing  $AS_1$  and opening  $S_1$  we get the ammeter reading for that particular cell. Closing  $VS_1$  we get the voltmeter reading. The terminals  $C_1$ ,  $C_2$ ,  $C_3$  are those of the Guess-Haultain apparatus with very slight modification, as shown in Fig. 5. These were slightly more satisfactory, due to the fact that our electrodes had much thicker handles than those used with the apparatus mentioned. The voltmeter and ammeter readings may be obtained in the same manner for any cell when any number of cells are in operation.

#### Process for Lead Ores.<sup>3</sup>

Since it is possible to remove the sulphur from galena as well as from some other sulphides, in the form of  $H_2S$  by warming with  $HCl$  (2-1), it is convenient to use the following method for those galena ores containing no interfering sul-

<sup>1</sup>Stoddard, J. Am. Chem. Soc., 31, 385 (1909); Palmer and Palmer, Trans. Am. Electrochem. Soc., 15, 489 (1909); Benner, J. Ind. and Eng. Chem., Vol. 2, No. 5 (May, 1910); Benner, Ibid., Vol. 2, No. 8, (Aug., 1910); Frary, 17th Gen. Meet. Am. Electrochem. Soc., Pittsburgh, May 5-7, 1910.

<sup>2</sup>Palmer and Palmer. Ibid.

<sup>3</sup>J. Ind. and Eng. Chem., Vol. 2, No. 8.



phides. Weigh out from 0.5 to 1.0 gram of finely pulverized ore, warm with 15 to 20 cc. HCl (2-1) until completely disintegrated and all of the sulphur evolved as hydrogen sulphide, but not to such concentration that lead chloride crystallizes out, causing bumping. Then add 20 cc. concentrated  $\text{HNO}_3$  and boil until all of the HCl is expelled and the nitric acid has a volume of about 10 cc. Dilute to 75 cc. and electrolyze in the usual manner. Those ores containing sulphides which render the preceding method impossible are disintegrated with HCl and  $\text{HNO}_3$ , then treated with 15 cc.  $\text{H}_2\text{SO}_4$  and the solution evaporated until heavy white fumes of sulphuric anhydride are evolved. After cooling, 50 cc. of water are added

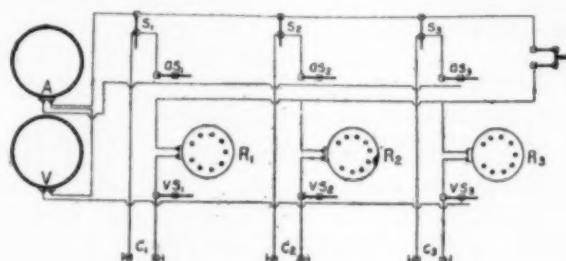


FIG. 4.—ARRANGEMENT OF AUTHOR.

and the solution boiled until all of the basic iron sulphate is dissolved. Dilute the solution to 20 cc., filter and wash free from sulphuric acid, washing by decantation as much as possible. The lead sulphate is then washed, with the smallest possible amount of water from the filter into the beaker containing the bulk of the sulphate washed by decantation. Pour 10 cc. of a warm saturated solution of ammonium carbonate through the filter, receiving it in the beaker containing the sulphate of lead, and warm until all of the sulphate is converted to the carbonate. The lead carbonate is then filtered through the same paper, washed with water until free from ammonium carbonate, dissolved in 40 cc.  $\text{HNO}_3$  (1-3), the filter paper washed free from lead, and the solution electrolyzed in the usual way. The following results on a lead ore containing copper show the accuracy of the method.

Analysis of the ore by ordinary methods: Ore No. 1, 2.7 per cent copper and 21.5 per cent lead; ore No. 2, 2.28 per cent copper and 22.2 per cent lead. Electrolytic determination of lead: No. 1, 21.52, 21.51, 21.59; No. 2, 22.2, 22.06 per cent.

Where it is desired to do away with the use of the factor, and at the same time use the electrolytic method of determination, it is possible to do so by igniting the gauze electrode on which the peroxide of lead has been deposited, in a muffle or small electric oven at a temperature of about  $500^\circ\text{C}$ . The following table shows the accuracy of this modification of the electrolytic method when a C. P. lead solution is used:

Lead taken.	Lead found.
0.2509	0.2511
0.2509	0.2506
0.5018	0.5035
0.5018	0.5026
0.7527	0.7518

The lead peroxide can be readily removed from the gauze electrode by means of dilute nitric acid containing a small amount of potassium nitrite.

#### Process for Copper Ores.\*

Dissolve from 0.5 to 1.0 gram of the ore, containing none of the interfering elements, in nitric or nitro-hydrochloric acid. When aqua regia is used, or if lead is present, sulphuric acid is added and evaporation continued until the heavy white fumes of sulphuric anhydride are evolved. The excess of sulphuric acid is neutralized with ammonia, and 3 cc. of concentrated nitric acid added. Providing no sulphuric acid is added it is necessary to evaporate to only 3 cc. In either case the solution is diluted to 75 cc., transferred to a tall battery

beaker without filtering (if the residue does not settle readily it is better to filter), and the copper deposited by means of a current of 5 to 8 amperes and 3 to 4 volts. If the deposit is black or dark colored it is dissolved in 3 cc. nitric acid and re-deposited in the same way. This redeposition is, as a rule, easier and quicker than filtering. Results are as accurate as with the older method if proper care is taken.

#### Process for Copper Slags.

In case of slags and other low grade material it is necessary to select a method of disintegration suited to the particular material in hand. Slags, which always give large quantities of gelatinous silica and contain small percentages of copper, give the most accurate results when disintegrated with nitric acid, followed by hydrofluoric and sulphuric acids. After evaporation until heavy white fumes are evolved the analysis is finished as for ores. Redeposition is more frequently necessary with low grade material than with high, but in all cases it is more accurate and rapid than filtering.

Elements which interfere, and at the same time are likely to occur in copper ores, are arsenic, antimony, lead and bismuth. The separation of antimony and bismuth from copper was tried under the preceding conditions, but without success. Arsenic in small quantities will not be deposited with the copper, but in amounts about equivalent to the copper present begins to be deposited with the last of that metal. This can easily be removed by dissolving the deposited copper and re-precipitating under the same conditions as before.

The removal of the interfering elements by chemical means can be accomplished by the following method: The ore is

disintegrated with nitric and hydrochloric acids, and then treated with 5 or 6 cc. of sulphuric acid and evaporated until heavy white fumes are evolved. Cool and dilute with water to about 150 cc. boiling to insure complete solution of the basic sulphate of iron, and adding a few drops of hydrochloric acid if silver is present. Filter

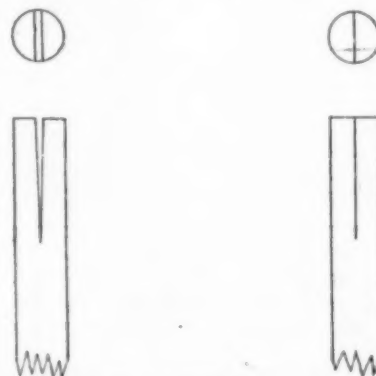


FIG. 5.—TERMINALS.

and wash with water containing a little sulphuric acid. The filtrate is now heated to boiling and saturated with hydrogen sulphide, allowing the solution to cool during the process. The precipitated copper is filtered and washed with water containing hydrogen sulphide. Rinse the precipitated sulphides from the filter paper into a beaker, using as little water as possible, and warm with a little colorless sodium sulphide to remove antimony and arsenic, and again filter through the same paper, using wash water containing sodium sulphide. Bismuth is now removed from the copper by warming the mixed sulphides with a solution of potassium cyanide made alkaline with ammonia. This dissolves the copper, leaving the bismuth. Filter, and boil the copper solution with nitric acid to destroy the cyanide, and electrolyze in the usual manner.

#### Copper Ores Containing Large Amounts of Iron.

When copper ores contain large amounts of iron it is impossible to deposit the copper successfully from a nitric acid electrolyte, as frequently before the deposition is completed a resolution of the copper often takes place from the combined action of the iron salts and the nitric acid. This phenomenon is well described by Classen in his work on electrochemical analysis, as observed by him and others.<sup>7, 8</sup> "Three grams of

\*J. Ind. & Eng. Chem., Vol. 2, No. 5.

<sup>7</sup>Fairlie and Boon. *Electrochem. & Met. Ind.*, Vol. 6, p. 58.

<sup>8</sup>J. Am. Chem. Soc., Vol. 32, No. 10.

a sample of copper containing from 40 to 50 per cent iron were taken, dissolved, diluted to 150 cc. and electrolyzed in the presence of 3 cc. of nitric acid, with a current of three amperes. As the precipitation of the copper began the solution took on a brown color. As it proceeded suddenly the solution was decolorized together with the solution of the copper, after which the electrolyte gave only a slight test for ferrous iron."

"This is explained as follows: During the electrolysis the hydrogen liberated reduces the iron to ferrous form, which with the decomposition products of nitric acid, forms the unstable compound  $x\text{NOyFeSO}_4$ . As long as the reduction of the iron takes place the precipitation of the copper continues, but as soon as this reaction reaches an equilibrium under the existing conditions, the unstable compounds suddenly decompose with the resolution of the copper and the oxidation of the iron, which is indicated by the decolorization of the solu-

as frequently they give high results. This may be due to the oxidation of the copper, or to the presence of interfering elements.

4. Use great care in adjusting the electrodes so that the current density is as nearly the same as possible at all points. When the density is higher at some points than at others it is likely to cause decolorization of the copper at these points.

5. In selecting gauze electrodes choose those which will allow the best circulation of the electrolyte and as large a surface as is convenient.

6. Under some conditions the deposited copper is likely to be spongy but will give good results if the deposit is not brushed off. Handle the electrodes in this case with the same care used with those on which lead peroxide has been deposited.

7. Do not try to deposit more than 0.2 to 0.3 gram of cop-

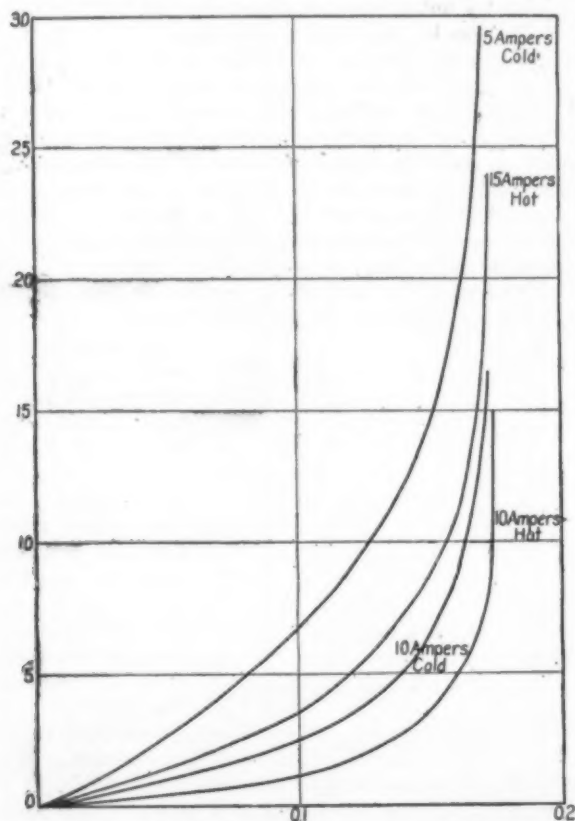


FIG. 6.—DEPOSITION CURVES FROM HOT AND COLD SOLUTIONS.

tion." Blaisdal and Harrison<sup>8</sup> have shown that it is possible to prevent the formation of nitrous acid and consequent solution of copper by the addition of 5 grams of urea to the acid solution of copper before electrolysis.

Figs. 6 and 7 give the rate at which lead and copper are precipitated with the electrode shown in Fig. 2 and under the conditions used for the precipitation of these elements in ores. Each point determined on the curves was made as a separate determination with a new solution containing the same amount of copper or lead in each case.

The following points may well be observed in the electrolytic determination of these metals:

1. Always keep the cathode completely covered by the electrolyte.

2. Care must be taken not to stop the electrolysis until the precipitation is complete. The time of deposition varies with different electrodes, volume of solution, amount of metal in solution, etc., even if the current is the same in each case. These facts are especially true with rapid methods and with the gauze electrode in particular.

3. Dissolve and redeposit all dark colored copper precipitates,

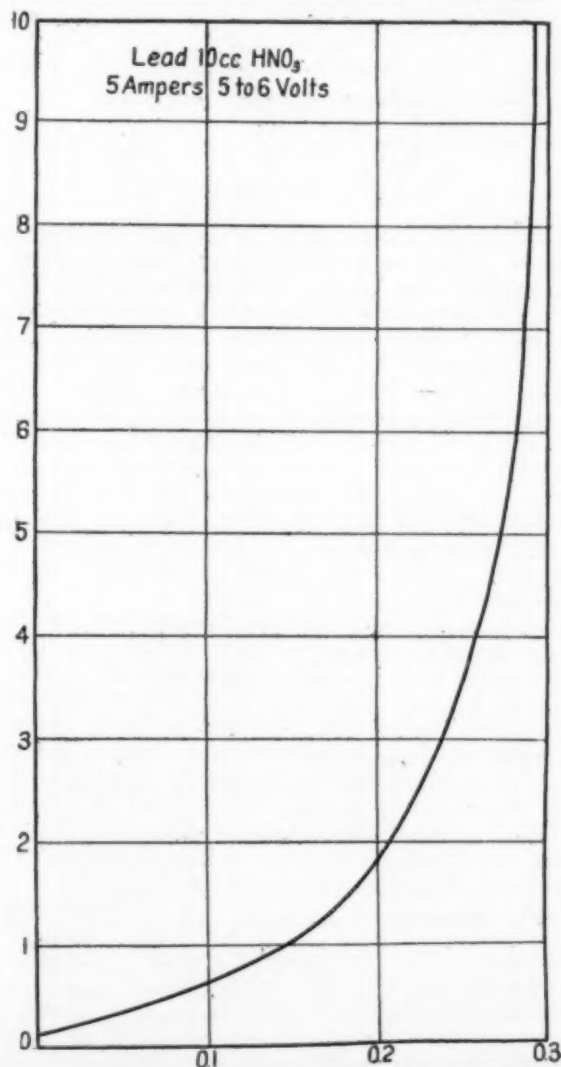


FIG. 7.—DEPOSITION OF LEAD. (ABSCISSAS REPRESENTING GRAMS  $\text{PbO}_2$ ).

per or lead, as such deposits are not as good as when small amounts are used.

8. Do not allow the electrolyte to boil away, as the concentration of the acid is likely to cause the solution of the metal.

University of Arizona, Tucson.

**American Society of Mechanical Engineers.**—The sixty-third meeting of the American Society of Mechanical Engineers will be held in Pittsburgh, Pa., from May 30 to June 2. This is the first meeting of the society in Pittsburgh since 1884. Col. E. D. Meier, of St. Louis, is president this year.



### Chemical Engineering in the Manufacture of Metallic-Filament Lamps.

It is unnecessary to emphasize at this place that the progress in modern lighting methods has been rendered possible essentially by the work of the chemist. This is true both for electric incandescent and arc lamps as well as for gas lamps.

In the manufacture of incandescent lamps the adventure of the metallic-filament lamp (first osmium, then tantalum, finally tungsten) was revolutionary after the carbon lamps had

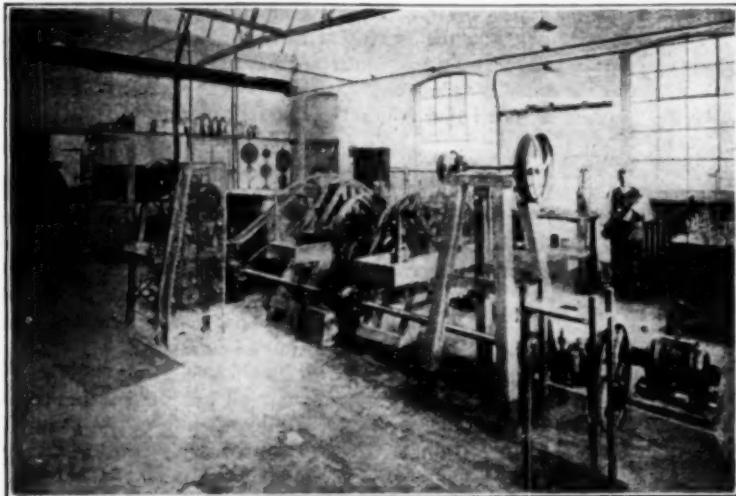


FIG. 1.—CHEMICAL DEPARTMENT, SHOWING MECHANICAL MIXERS IN FOREGROUND.

reigned supreme for so many years. In the preparation of the tungsten filament chemical engineering plays a large part in all the different processes devised for this work. Undoubtedly the most important recent advance is the method developed in the research laboratory of the General Electric Company under the direction of Dr. W. R. Whitney for producing ductile tungsten, as noted in an American Electrochemical Society paper of one of the inventors, Dr. C. G. Fink (our vol. VIII, p. 340).

Nevertheless the older methods of making tungsten filaments are still of great importance and considerable interest and in view of the secrecy which generally surrounds metallic-filament lamp works some notes on a British tungsten-lamp factory—the Brimsdown Metal Lamp Works—are particularly interesting. They are taken from the *London Electrician* of Jan. 27 and *London Electrical Engineering* of Jan. 26.

The process of Dr. Kuzel in which the colloidal state of tungsten is made use of is employed in this factory. The filaments in the Brimsdown metal lamps are produced from colloidal tungsten, and not from tungsten powder. The colloid is mixed with water in the chemical department, and the solution is placed in bottles which are clamped securely in frames and rotated slowly through gearing by a small motor. These frames are shown in the foreground in Fig. 1. The process is one that ensures the thorough mixing of water and the colloidal tungsten.

The mixture is then reduced to a plastic state by another process and passed between water-cooled rollers. From these rollers it emerges as an extremely thin sheet of metal, and is twisted up and is pressed into a cartridge.

This cartridge is passed into the squirting department, a general view of which is given in Fig. 2. The actual squirting of the filament is carried out both by hand pressure and hydraulic pressure. In the illustration (Fig. 2) three hydraulic presses will be noticed on the right and six hand presses on the left. The material is squirted through diamond dies, which vary in diameter according to the candle-power and voltage of the lamps for which the filaments are intended. The filaments for the lamps of ordinary size are finer than a human hair, ranging from about one-fortieth to a millimeter upward in diameter.

The squirting is carried out vertically downward, and the thin material is received on a large white card, which is moved backward and forward so as to spread the filament over the card. When the card is full it is passed on to benches in another part of the same shop, and in the soft state the filaments are divided off into U-shaped loops. They are then laid in metal trays, which are placed in baking ovens. In these ovens the moisture is driven off and the filament loops are then cut off to standard lengths.

The forming department of the works is one of the most interesting, because it contains a quantity of special apparatus which has been designed and manufactured by the company's staff. This apparatus is fitted up in units, each of which is complete in itself, and mounted in groups on the benches, which run transversely across the shop.

The filament-forming process consists of a chemical and electrical treatment of the filament under a glass bell. The U-shaped filament is held in two clips, which serve both as supports and terminals for the testing circuit. A tungsten wire is hung at the bottom of the U as a weight. The bell is then partially exhausted and ammonia gas is admitted in a certain quantity into the bell. An exhausting tube is also provided to extract the air and other products during the forming process. Current is applied to the filament, and the voltage is gradually raised from practically nothing to 50 per cent above normal. This is controlled absolutely automatically by a motor, so that the current is gradually increased up to a certain value and then switched off. During this process the filament changes



FIG. 2.—VIEW OF FILAMENT SQUIRTING SHOP.

its form materially due to the weight of tungsten wire attached to it.

All the binding material is driven out of the filament and only the tungsten metal remains. The entire operation is automatic after the bell has been lowered, and four filaments are treated at a time consecutively. All the operations are carried out by girls, who, of course, have no technical knowledge and do the work entirely mechanically. Further, the

quality of the product is uniformly good. This indicates the ingenuity of the automatic apparatus.

After the filaments are removed from under the forming bells they can be taken by the ends and twisted backward into a circle, a test which speaks highly for their strength and flexibility.

The forming benches occupy the bulk of the space in the filament factory; 80 girls are employed at the forming benches, each of whom can form a maximum of 600 filaments a day. At one end a number of motor-generators and air and ammonia compressors have been installed. At the other end of this department is a separate section, where the filaments are weighed and carefully cut off to standard lengths on a gauging and cutting machine. The weighing machines will register up to a tenth of a milligram.

Arrived at this point in the process of manufacture, the filaments are ready for mounting, but this work is carried on in a separate building. The lamp supports are made, the filaments are mounted in position, they are then welded in place, placed in the bulbs, and the final process of exhausting and sealing in is gone through. All these operations are carefully subdivided among the employees into groups which practically form separate departments in themselves.

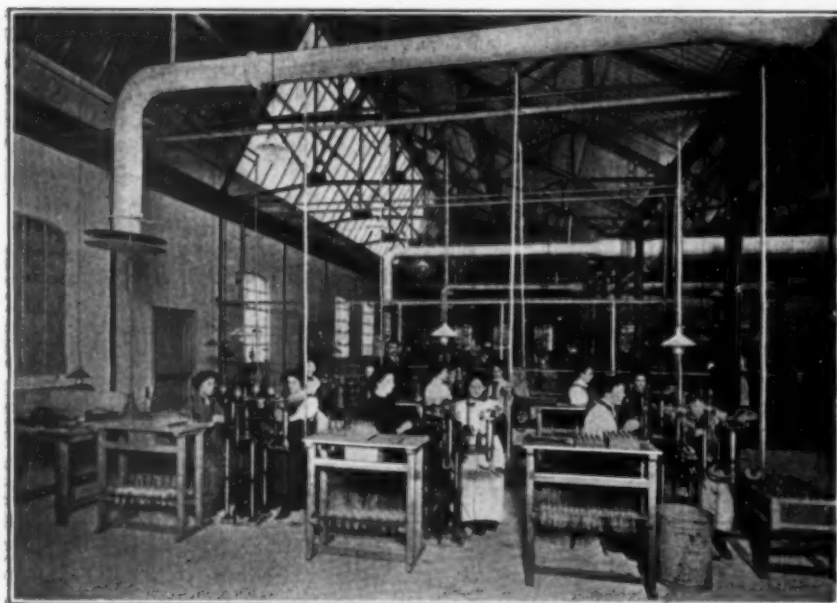


FIG. 3.—SECTION OF GLASS-BLOWING DEPARTMENT.

The foot-making, stem-making, fusing into the glass of the leading-in wires, welding of platinum tips, the fixing of the top and bottom hooks and supports for the filament, are several delicate operations which are entrusted to each group. As the work proceeds in each department it is passed into a checking section, carefully examined and tested and passed out again, when approved, to the next section.

The leading-in wires are formed of short lengths of copper, platinum and nickel wire welded together, the platinum being a short length to pass through the glass. It is stated, however, that a new material (alloy) is being introduced in place of the platinum, which has, as platinum, the same coefficient of expansion as glass. The leading-in wires are inserted in their glass tube, which is pressed round them after being softened in a blow-pipe flame, the central glass stem is made with its circle of thin and flexible nickel hooks at the other end, and the filaments are placed in position and pasted to the lower nickel wires with a paste of finely divided tungsten paste.

The work is then checked and the filament cage goes to the welding section, where the pasted joint is electrically welded in an atmosphere of hydrogen under a glass bell. In this sec-

tion the work is again checked and issued, after approval, to the sealing-in department, in which the filament cages are fused into the lamp bulb. This work is carried out on semi-automatic machines which are also attended by girls.

The final process, previous to the actual testing of the lamp, is that of exhausting and sealing off. This is carried out at special benches, each of which is fitted with a unit apparatus comprising a heating box to warm up the bulb, a number of vacuum nozzles, switches for putting the lamps in circuit, and a blow-pipe flame for sealing off.

The initial vacuum is obtained by electrically driven vacuum pumps, which are in operation in a building adjoining the lamp department, and the final exhausting is carried out by Gaede rotary mercury pumps which are placed immediately adjacent to the exhausting nozzles. The outfit is the latest of its kind, and it is stated that it ensures a nearly perfect vacuum.

### Manufacture of Aluminium.

In an article by Mr. WM. CLACHER in London *Electrical Review* of Jan. 20 some details are given on the manufacture of aluminium as carried out by the British Aluminium Company, Ltd. His notes on the manufacture of carbon electrodes by the same company are dealt with in a separate article in this issue.

From the molten electrolyte, consisting of alumina and its flux cryolite, there is reduction of the oxide at the point of contact of the carbon anodes, the carbon oxidizing and burning away. The metal thus produced sinks to the bottom of the electrolytic furnace, which is carbon-lined, and is drawn off thence.

This metal, from the purity of the material used—the carbon being 99 per cent and the alumina 99.9 per cent purity—is of very high percentage; over 99.5 per cent is very usual.

From the aluminium reduction works the ingot metal is taken to the mills and foundry to be worked into sheet and alloy and also for castings.

The 10-in. square carbon blocks are arranged in sets of 12 in the electric furnaces. The direct current for operating the electrolytic furnaces is obtained from 50-volt dynamos.

The plate for the bottom of the furnace is of cast iron and somewhat of the form of a grid. The interspaces are filled

with carbon which is rammed and the whole plate covered. This form makes the contact between the carbon lining of the furnace and the cable much more effective.

The resistance of the carbon blocks is tested at the furnace. The carbon blocks are consumed away, the gases going to the atmosphere; hydrofluoric acid gas is also evolved from the cryolite used.

The metal as it comes from the furnace is afterward melted and the pure metal, free from any mechanical admixture, is molded in ingots. These ingots are bagged, about 20 ingots to the bag, and dispatched to the foundries and mills.

An allocation of the cost of producing the metal is given by Mr. Clacher as follows:

	Per Ton.
Energy .....	\$25.00
Carbon .....	32.50
Alumina .....	140.00
General expenses and profit.....	140.00
	<hr/>
	\$337.50

This allocation of the cost seems interesting.



The power used is water-power, the most economical source of energy.

The alumina is purified from the bauxite by the Bayer process—solution in caustic soda and precipitation therefrom by addition of some previously precipitated alumina, filtered off and kilned.

The bauxite is first kilned to expel moisture, then finely pulverized; the finely divided material is mixed with caustic soda of 40 per cent strength to a fluid mass and kied under an internal pressure of 30-60 lb. per square inch, 100 lb. steam jacketing the vessel. After eight hours the mass is blown by its internal pressure to a tank, whence the filter presses are charged; the height of the tank above the level of the presses depends on the fineness of grinding. The insoluble matter is termed red mud.

The mass when blown to the top tank is diluted to a suitable concentration and temperature, so that the wear and tear on the cloths is not heavy.

The filtered aluminate of soda is run to tanks and some alumina, previously precipitated, is added. After some time the alumina is precipitated, a white semi-flocculent substance.

This precipitate is then filtered, washed, partially dried in the press and calcined in a furnace at a cherry-red heat, after which the hard white amorphous alumina is ready for transportation in bags to the electric furnaces.

In analyzing the metal the impurities alone are determined, the difference being assessed aluminium. Of these impurities the most important are silicon, silica, sodium and iron. In practice the items customarily determined are the iron, and silica with silicon.

#### A New Machine for the Separation of Liquids and Solids by Centrifugal Force.

Centrifugal force is the keyword of the many inventions which have made the name of De Laval famous in the past thirty years. This is as much true for the big De Laval steam turbine which has played such an important part in the mod-

ernization of power plants as for the small De Laval cream separator which is described and illustrated on page 159 of this issue, has been in successful commercial operation for some time for the clarification of liquids containing a small amount of sediment, etc.

But the bigger problem of separating liquids from solids in general (say mixtures containing from 2 per cent to 20 per cent of solids) by centrifugal force has only recently been solved. The new centrifugal separator, about which the first advance information is given in this article and which has not yet been placed on the market, is the outcome of ten years of continuous work of the engineering experts of the De Laval Separator Company, in the course of which some twenty-nine patents have been taken out for principles and details. But in view of all this, really the most surprising and most characteristic feature of the whole is the simplicity of the design.

The diagram, Fig. 1, gives a section of the machine. An outside view is given in Fig. 2, which shows a machine installed in the experimental plant. This does not represent ideal conditions, so that for continuous operation on a commercial scale some of the arrangements around the machine would be changed. The separating machine itself has so far proven equal to any task, even the most difficult, to which it has been put. Fig. 3 shows the feeding end of the machine and Fig. 4 the opposite end, with cover or cap removed, which shows the differential gear-driving mechanism. All of the moving parts of the machine are carefully encased, the only exposed part being the drive pulley, which is mounted on the shaft shown in Fig. 3.

The working or enclosed cylinder is 30 in. in diameter by 8 ft. long and revolves at a normal speed of 320 r.p.m., equal to a peripheral speed of 2513 lin. ft. per minute. Thus a product delivered into the machine is subjected to a centrifugal pressure, which causes the heavier part (the solids) to settle to the periphery.

The operation is as follows:

The raw product is delivered at *A* (Fig. 1), into a hollow arm of the supporting spider, through which it passes to the revolving cone *B*, thence to *C* and *D*. During the entering

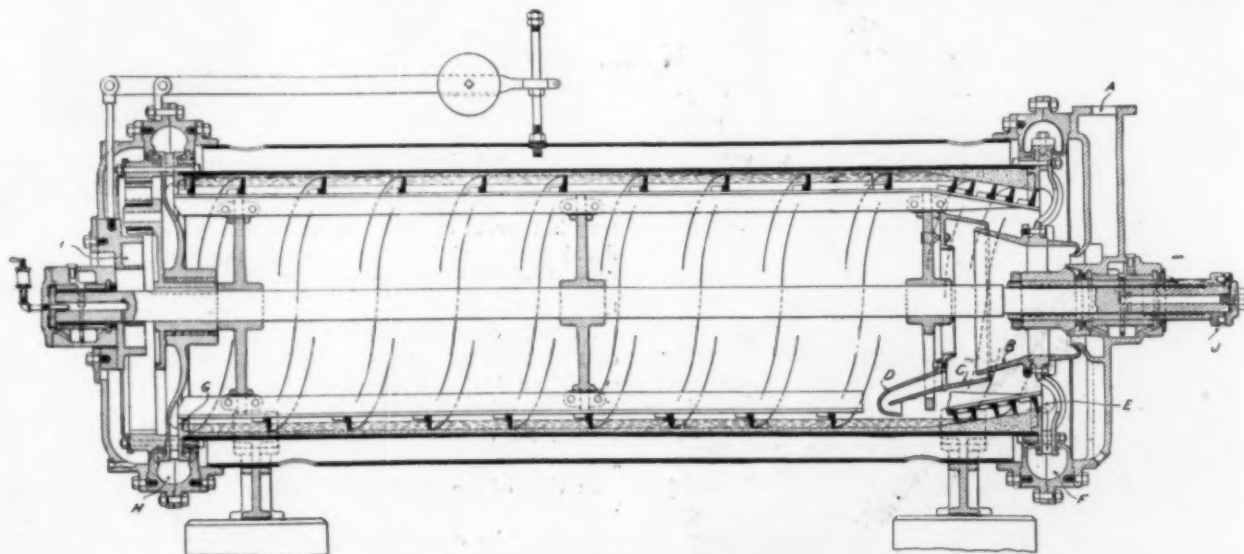


FIG. 1.—DIAGRAM OF CENTRIFUGAL SEPARATOR.

ernization of power plants as for the small De Laval cream separator which has completely revolutionized the dairying industry.

The industrial applications of centrifugal force are almost limitless, and it is no wonder that the De Laval concerns are constantly working out new machines for new purposes. In the chemical and metallurgical industries the use of centrifugal force suggests itself naturally for the separation of liquids and

process the raw product attains practically the peripheral speed of the machine. The coarser or heavier solids are at once separated from the liquid, and by a series of broken helical scrapers, operating at a differential speed, they are drawn up an incline or cone, freed from the liquid, and discharged by the centrifugal force at *E* into the receiving chamber *F*. The liquid is discharged at the opposite end at *G* into the receiving chamber *H*.

The liquid in passing to the discharge end of the cylinder travels a tortuous course, equal to 27 lin. ft. in distance, which may require one or more minutes to pass through, thus subjecting the lighter, or difficult to settle, solids to the high centrifugal pressure for a much longer period than if it was

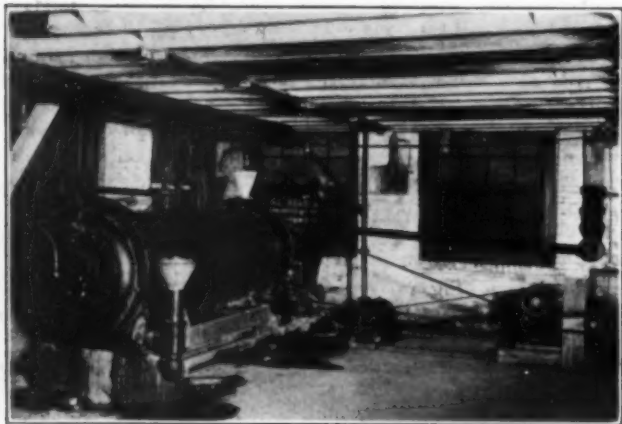


FIG. 2.—OUTSIDE VIEW OF CENTRIFUGAL SEPARATOR.

passed straight through the cylinder, the advantages and importance of which will be fully appreciated. During this period the solids are practically all settled to the periphery of the revolving cylinder, and as there accumulated from each point they are slowly drawn the length of the cylinder by the scrapers to the inlet end and discharged at *E*, as above described.

The capacity of the machine varies according to the kind of products, but unless very difficult the capacity ranges from 2400 to 3000 gal. per hour.

The speed of the machine may be varied from 150 to 350 r.p.m. to accommodate the different kinds of material.

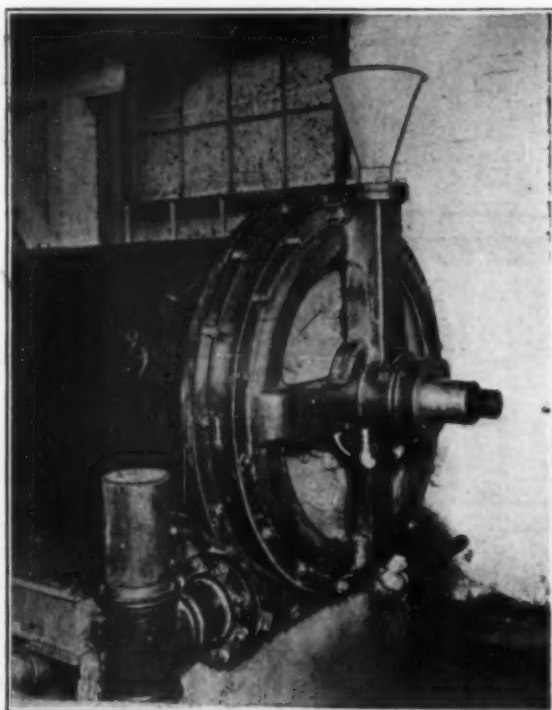


FIG. 3.—FEEDING END OF SEPARATOR.

The lever shown on the top of the machine with the round weight on end (Fig. 1) is the safety device connected to the differential driving gears and which prevents possible damage to the machine in the event of overloading or other cause.

The lever is adjusted to permit of a certain pressure or force on the scrapers, and when it exceeds this amount the lever automatically rises, relieves the pressure and throws the differential driving mechanism out of commission, causing the scrapers to run in unison with the cylinder and as one piece. Without going into details, it may be said that this is one of the most ingenious details of the machine.

The machine when once adjusted to a product can be run continuously.

The weight of the machine complete is approximately 7000 lb. The oiling is largely automatic, and one man could manage and care for a number of machines.

This separator has now been working for 2½ months, for nine hours a day, in the experimental shop on a large variety of products, of which starch, sand, clay and whiting may be mentioned. Other materials are to be experimented with; thus the separator will be tried for cyanide work, etc.

From the samples of separated materials which we have seen it is safe to say that the separation obtained with this

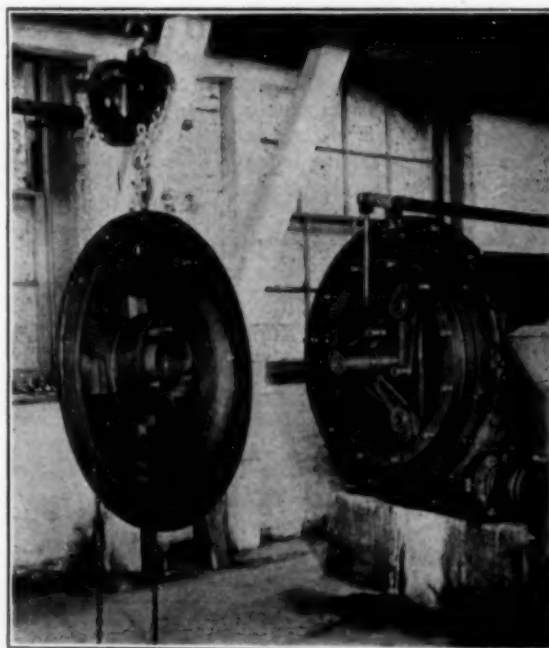


FIG. 4.—DIFFERENTIAL SPEED MECHANISM.

machine is practically perfect. If it is desired to remove the very last traces of sediment, which may have remained in the liquid discharged from the separator, this liquid may be passed through a centrifugal clarifier and filter, as described on page 150 of this issue.

Mixtures combining from 2 or 3 per cent up to 15 or 20 per cent of solids in water may be treated by this machine. The power for driving is very low. On a mixture containing 12½ per cent solids less than 5 hp is required. As mentioned before, the capacity of the machine ranges in general from 2400 to 3000 gal. per hour, according to the kind of material.

In view of the simplicity of design and practical fool-proofness of the machine, the cost of attendance and operation is very low. Its compactness and efficiency of handling quickly large amounts of difficult material should certainly attract the attention of the chemical and metallurgical engineer.

When the machine is placed on the market we expect to give further descriptions, with results of tests, though in the meantime we are sure the manufacturers will be pleased to give interested parties such further information as they may desire.

This new De Laval centrifugal separator is certainly an interesting example of the possibilities of centrifugal force and of the effectiveness of co-operation between the chemical and mechanical engineer.



### Some Practical Experience with Corrosion of Metals\*

BY REAR-ADMIRAL JOHN T. CORNER, C.B.

It should be pointed out at the outset that in this paper the writer purposes to limit himself to chiefly giving instances of corrosion which have come under his notice while actively engaged as a marine engineer and to detail some of the practical methods found to be efficacious in dealing with the same.

Some of the causes of corrosion of metals on shipboard are so obscure, and the origin so difficult to trace, that no satisfactory explanation is so far forthcoming. On the other hand, cases have occurred in which the causes operating to produce the effects are clearly recognized and easily dealt with.

The writer has had no personal experience of the trouble which took place through the corrosion of copper and its alloys in the old wooden ships, but that such corrosion did exist is within the knowledge of most of those who have studied the record of such ships. Cases are mentioned of copper sheathing having become corroded in mysterious ways and at unexpected times, and there are reports of the heads of the nails having corroded entirely off, allowing the sheathing to become detached.

There were doubtless at times troubles through corrosion in the bilge and other copper pipes of the old wooden steamships of war, but as a rule these appear to have been of minor importance; and in many ships the chief anxiety the engineers had concerning the pipes was to keep them in a bright and burnished condition so as to pass the captain's weekly inspection, at which the array of copper piping in the bilges was often one of the show sights of the ship.

With the introduction of iron for shipbuilding purposes the conditions are different, for it was soon found necessary to insulate as well as protect the copper pipes in or near the bilges—not, however, for their own welfare so much as for the sake of the adjacent plates and angles, which, when not so treated, were found to be adversely affected by contact, owing to the galvanic action set up between the metal pipes and the iron plating by the bilge water.

There were many instances in which very considerable trouble was caused. Among them may be mentioned the case of the troopship *Megaera*, which vessel had to be beached at St. Paul's Island about 1869 to prevent her sinking. It came out in the subsequent inquiry that among other defects in this ship there was a serious one due to a copper strainer fitted to a bilge section pipe in a remote part of the ship. The action set up by it was sufficient to eat right through the plates and so admit water to such an extent that the ship had to be run ashore to save the crew.

As far as can be ascertained the pipes themselves in iron ships were no more affected than those in wooden ships, probably not so much; and it was not until at or about the time of a great change in the internal economy of men-of-war that serious and persistent troubles through the corrosion of copper pipes began to be manifest and felt. This change was the introduction of the electric light on shipboard.

The writer's first experience of the insidious corrosion of copper pipes occurred in a ship which had recently been fitted with an electrical installation for working searchlights. During a commission of over three years there were, as far as can be remembered, only two pipes which seriously suffered from corrosion; both were water pipes, and both were eaten right through in small holes, which had the appearance of having been punched, and it is noteworthy that each of these pipes was in the immediate vicinity of an electric lead to a searchlight—the one forward and the other aft. The whole business was after all but a minor affair, and consequently very little notice was taken of it; it was only years afterward, when trying to assign a cause for similar trouble in various other ships,

that the previous experience was recalled to throw light on an obscure question.

There is no doubt whatever that the time of the construction of the electric light in naval ships synchronized with a considerable increase in the corrosion of the copper and metal pipes and fittings; and as the parts affected were chiefly water fittings, the increased steam pressures which came at or near this period, and which might have been suspected as one of the causes, can be now quite absolved from any share or part of the trouble.

Sometimes pipes would be found eaten nearly, and in some quite, through in a very short period; moreover, the replaced parts often became similarly affected even when different brands or qualities of copper or brass were used. Sometimes the pipes were eaten in holes having sharp, clean-cut edges, as though punched; sometimes parts were pitted and honey-combed over a considerable area of surface; while in other examples the metal was nibbled out in such a way as almost to suggest the presence of rodents or microbes.

As regards the source of this trouble, some engineers for the reasons above stated were inclined to attribute it to electrical action due to the leakage from the mains, which was often so great as to allow electricity to permeate the whole ship. It must be admitted now that some of the earlier electrical installations on shipboard were a long way from perfect, inasmuch as leakage in some cases was sufficient to turn the ships themselves into storage batteries. Others attributed these defects in the copper pipes to induced currents set up in different parts of the ship by the currents in the electrical mains. But nothing definite could be proved, and those then dealing with the electrical plant disclaimed all responsibility for the trouble, and in fact sometimes expressed the opinion that the currents they used could hurt nothing and cause trouble to nobody.

It was also further suggested that the action might be due to mere chemical causes, but here again the responsibility for this was disclaimed by those concerned, the corrosion being put down by them to mechanical rather than chemical action; and there was certainly some ground for the contention, seeing that corrosion in pipes generally occurred at or near bends or branches; also very extensive corrosion often occurred on the bronze propeller blades, especially at the forward side where there was little likelihood of extensive action, electrical or chemical, but abundance of the mechanical kind.

It is very probable that a certain amount of chemical action was first induced by galvanic action, and the erosion caused by the friction of the water, which washed away the results of the chemical action, left the surface exposed to further and similar action. This explanation, however, would not fully apply to cases where clean-cut holes were found in the pipes, condenser tubes, etc.; these results were probably due to want of homogeneity in the composition of the metal, the galvanic and chemical action finding out the weakest places. It would also appear that the presence of air and moisture together largely influences the extent of the corrosion of these metals as of iron, and in a measure helps to explain the pitting on the forward sides of the propeller blades, where the partial vacuum formed by the motion of the screws causes air to be drawn from the surrounding sea water, which assists the corrosive action.

It was also suggested that the corrosion might be due to the use of copper of too pure a quality; and in this connection it may be noted that some locomotive engineers specify for about ½ per cent of arsenic in the copper fire-box plates, as a preventive against corrosion. So there really may have been something in this suggestion, seeing that pure electrolytic copper was beginning to be used at or about the time referred to.

But whatever may have been the causes, the pressing business of the day was to discover the cure of the complaint; the prevention of the corrosion, which affected most of the water pipes, including the condenser tubes of many ships, was what all aimed at. As regards the latter, it is understood there was

\*A paper read before the (British) Institute of Metals at the January, 1911, meeting.

not much general corrosion of the tubes in iron condensers. No doubt there have been, as there still are, some instances in which the tubes of iron condensers have given considerable trouble through various causes, some of which were foul circulating water, defective materials, excessive vibration through indirect local pressure, such as the direct impinging of the steam or circulating water on the tubes, etc.; but as regards the corrosion under consideration, it may be stated generally that it is certainly much less in condensers with iron shells than in those with brass shells.

One special case of an iron condenser which came under the writer's notice was interesting and unique, and may be mentioned here: A ship with a cast-iron condenser, with circulating water outside the tubes, had been running for several years with a set of tubes which had given no trouble whatever. The ship unfortunately got ashore on a sandbank, and in getting off stirred up the sand so much as to get a large quantity deposited in the condenser on the circulating sides. On returning to port for overhaul and repairs, the tubes were taken out to get rid of the sand. This afforded the opportunity for examining and attending to the inside of the condenser shells. They were cleaned by being chipped and scraped, and finally given a good coating of red-lead paint. Shortly after the ship resumed service the condenser tubes, which up to then had given no trouble, began to go to such an extent that the ship had to be again laid up for the complete retubing of the condensers. When this was done the red-lead paint was all removed, after which no further trouble with the tube was experienced. This result clearly showed that the brass tubes were being protected when the iron shells were bare, but when covered with lead paint a corrosive action on the tubes manifested itself.

Another case of the protection afforded by iron may be mentioned: In a certain ship one particular pipe conveying circulating water was repeatedly in trouble through corrosion. This pipe would be found badly eaten away at a bend. It was taken down, patched and replaced time after time, but the corrosion still went on and continued. The chief engineer then resorted to the following expedient: He sweated a lump of copper on the outside of the pipe, so as to provide material for a screwed hole, through which a large iron bolt, about  $1\frac{1}{2}$  in. diameter, was fitted so that the point protruded; and no further trouble was experienced with this pipe, the bolt being screwed further in as the point became eaten away, and being entirely renewed when necessary.

These two examples appear to indicate that the presence of sufficient bare or unprotected iron in perfect metallic contact with the system strongly counteracts the corrosion of the copper or brass; the chemical or electrical or mechanical action, whichever it may be, solely or jointly, being entirely expended on the iron. It is therefore by the free use of iron or steel that it is considered the best protection of the copper or alloy pipes is provided.

Before resorting, however, to such free use of iron various other means were tried in different ships. One method was to coat the insides of the pipes with marine glue, another to use plumber's solder or tinman's solder as an internal coating. Special alloys containing aluminium were tried. Zinc slabs and blocks were used, as also zinc bars let in among the condenser tubes; and though some of these expedients effected a certain amount of good, it is considered that the present practice has more to recommend it than any of them, and that is the insertion of short iron lengths of pipe in runs of water pipes which can be easily removed and replaced; and further, the use of iron or steel doors for brass condensers with iron rods and plates in and about the tubes is very effective as a protection to the tubes and plates.

In some warships of late years the condenser barrels are made of steel. Of course in this latter case there is the possibility of a certain amount of corrosion in the condensers themselves, but as this is only to be expected on the water-

side, it should not be difficult to deal with—the design of the condenser being such as to admit of the easy replacement of parts likely to be affected and provision being made for zinc protectors as in boilers.

It is considered more than probable that the reason why iron is found on the whole to be more efficient as a protective than zinc is that with the former better metallic contact is assured and maintained, and perfect metallic contact is undoubtedly an absolute necessity.

Respecting the corrosion on propeller blades, it is not known to the writer whether iron has been tried as a protection against corrosion; but he would suggest for consideration that if easily renewable plugs of iron could be put at the parts of the propeller blades where corrosion is likely to occur they might lead to a marked difference in the extent of the pitting, for it may possibly be found that so long as there is iron in the immediate vicinity sufficient to absorb the corrosive action the bronze will be entirely protected.

### Notes on Chemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

#### The Corrosion of Metals and Alloys.

Mr. G. D. BENGOUGH's lengthy preliminary report to the corrosion committee of the Institute of Metals was discussed at the annual meeting of that body on Jan. 18.

Sir William Tilden was of opinion that iron in the metallic form undoubtedly afforded protection, yet the oxide of iron, if deposited locally, was a cause of corrosion in brass tubes he had examined a quarter of a century ago; and it should be noted that ferric oxide was electronegative. There were great differences of opinion on the subject of corrosion, and it was necessary that the question should be carefully investigated. He emphasized the necessity of assimilating the conditions of the experiments to those of actual practice.

Dr. W. Rosenhain said reference had been made to the dimensions of the microstructure as affecting corrosion and there was evidence that the metal having the corrosion was more liable to pitting. If two pieces of brass were taken, one lightly strained and the second more severely strained, a coarser structure was given to the latter. It was possible that similar differences in the microstructure might result from an unequal treatment of different parts of the same tube during the drawing process.

Mr. A. E. Seaton said that brass tubes were largely used in the navy for ships' boilers 30 years ago, with iron stay-tubes, and there was not any trouble from pitting then. Condenser tubes were formerly of 16-gage thickness, but were now of 18-gage to 20-gage.

Prof. T. Turner said he had conducted some experiments on the green deposit which formed on condenser tubes. The deposit was a non-conductor and was not part of an electric cell, but that fact did not controvert the view that the green deposit might play a part in causing corrosion.

Dr. J. T. Dunn thought that pitting was the result of the presence of local impurities introduced with the water. It had been proved that local corrosion of brass was set up by small pieces of rusty iron when immersed in sea water.

Mr. Arnold Philip (Admiralty) objected that the report had been made rather from the academical than the practical standpoint. A vast amount of useful information was available from manufacturers and users on the subject of corrosion and it was a fundamental omission not to have drawn more largely on their experience. The causes of corrosion were very numerous. The chief of these were the attacks of particles of carbon, etc., hydrochloric acid in the steam, externally applied e.m.f. mechanical abrasion by steam and water, presence of air and gases at bends of tubes, thermo-chemical action, mechanical strain, intrusion of foreign matter at surface of tubes, pres-



ence of two or more phases in the alloy, character and speed of the cooling water, and impurities in the alloy.

Mr. E. L. Rhead mentioned the influence of difference in temperature on the inner and outer walls of condenser tubes as a possible cause of accelerating corrosion. The question of annealing temperatures had an important bearing on the life of the tubes. With regard to crystalline structure, manufacturers employed photographs of crystals in different sizes in order to work within specified limits. He had recently examined some steel tubes which had corroded and found that failure was due to the use of an originally porous material.

#### Copper-Zinc Alloys.

Another paper read at the annual meeting of the Institute of Metals was on "A New Critical Point in Copper-Zinc Alloys: Its Interpretation, and Influence on Their Properties," by Prof. H. C. H. CARPENTER and Mr. C. A. EDWARDS.

The discussion was opened by Dr. W. Rosenhain, who, while admitting the existence of the critical point, was not able to agree with the authors in their interpretation of the facts.

Mr. G. D. Bengough was not able to coincide with Dr. Rosenhain's view, and, besides accepting the authors' theory, considered that the paper marked a distinct epoch in the history of brass. Recent researches of his own on the mechanical properties of alloys at high temperatures and on delta metal indicated that a remarkable change took place at the critical temperature, which, in his experiments, was  $510^{\circ}\text{C}$ . He also found a corresponding change in the 70:30 brass. His experiments on Muntz metal showed that at  $445^{\circ}\text{C}$ . there was not any B. constituent, which result was in accordance with the authors' conclusions.

Prof. A. K. Huntington remarked that the results of this research seemed to imply that with prolonged lapse of time all copper-zinc alloys would become brittle. He had lately made alloys with 36 per cent of zinc and less, and found that these—even after annealing—contained an appreciable proportion of the beta constituent.

Mr. E. F. Law said it was doubtful whether all inter-metallic compounds retained their identity when in solution, and this was a question which was well worth investigation. His opinion was that inter-metallic compounds were loosely combined and might behave similarly to double salts. The critical point dealt with by the authors was not a new one. Roberts-Austen recorded its existence in 1897, and, although Shepherd, Tafel and Bornemann ascribed his discovery to experimental error because they failed to detect the thermal change in question, the authors had now established beyond question that Roberts-Austen's observations were correct.

Mr. O. F. Hudson was not satisfied that the metal lost its brittleness above a certain temperature, and could subsequently revert to a brittle condition on further cooling. Brass was brittle only within a limited range of temperature. The authors' theories were consistent with the thermal observation, but not with the microscopic evidence.

Mr. E. L. Rhead said that calendering rolls, which were forced onto steel mandrils by heavy pressure, withstood a long period of work at high temperature, but after some months a longitudinal crack developed in consequence of a complete change in the character of the metal resulting from prolonged heating, and there was not much doubt that this structural change was connected with the point of inversion referred to by the authors.

Prof. T. Turner said that the establishment of this arrest point was of great consequence to brass manufacturers. It had long been known to practical men, but had never been mentioned in any text-books.

#### British Engineering Trade Returns.

The official statistics for 1910 published on Jan. 12 in the *Board of Trade Journal* show increases in both the imports and exports of all the principal engineering materials and manufactures. The imports of iron and steel amounted to £9,093,795, an

increase compared with 1909 of £1,122,201, while the exports reached £43,002,937, a rise of £4,810,795. The imports and exports of other metals and manufactures thereof were £24,699,531 and £10,360,159, the respective increases being £353,203 and £1,651,214. The imports of electrical goods increased by £363,960 and the exports by £1,887,041, the totals for the year being £1,686,469 and £4,117,840. The imports of machinery amounted to £4,471,303, an increase of £32,967, and the exports to £29,296,721, a rise of £1,239,078. The value of ships imported was £27,104 and exports were £8,769,331, increases of £3,178 and £2,842,217. The exports of ships decreased £1,798,144 from the values in 1908.

#### Production of Nitric Oxide by Combustion of Hydrogen.

At high pressures the combustion of hydrogen in a mixture of equal molecular parts of oxygen and nitrogen produces a considerable yield of oxides of nitrogen. A. WALOKITIN states that, without heating the gases, about three molecules of nitric acid in 100 of water—approximately ten per cent nitric acid—are obtained. Pressure exerts an appreciable effect only above three atmospheres, with a maximum effect at 16 atmospheres. With excess of oxygen practically the same results ensue. The yields agree with earlier results with explosion methods; but—probably in consequence of the theoretical temperature not being reached—are rather lower than the calculated values. The yield from the combustion of hydrogen in air is, of course, lower.

#### The Electric Furnace in Great Britain.

The *Times* of Nov. 30 calls attention to the reluctance of British steel manufacturers to advance with the times in the matter of electric furnaces. The article says that to England, with a rapidly diminishing stock of high-grade ore, the electric furnace is of peculiar value, as offering a means by which high-class steel can be manufactured from such low-grade ores as those from Cleveland, Lincolnshire and Northamptonshire; and proceeds to combat the argument that the absence of water-power is a fatal impediment to the extensive adoption of the electric furnace in England. The writer points out, too, that in the great manufacturing centers waste gases from blast furnaces and coke ovens as well as exhaust steam furnish sources of power which could be utilized for electric steel refining; and, further, that it is believed that there is at least one English manufactory which is buying electric energy at  $\frac{1}{8}\text{d}$  per unit. This is an exceptional case, but indicates future possibilities. The article referred to is inspired by the adoption of the Heroult patents by the United States Steel Corporation.

#### The Cost of Electric Energy in England.

An editorial note in the Engineering Supplement of *The Times* of Nov. 30 pointed out the misapprehension existing with regard to especial cheapness of electric energy obtained from water-power abroad as compared with the rates at which current can be supplied in England. In *The Times* of Dec. 7 Mr. A. S. Esslemont stated that he had recently made a tour of Norway, Sweden and southern Europe to study this question and found the average price of electric energy from water-power no lower than can be obtained to-day, for similar loads, from the power companies on the northeast coast of England. He pointed out that the very low prices often mentioned as being prevalent in Norway and Sweden are frequently misleading, the prices varying from \$6.25 to \$8.75 per electric hp-year. He stated that the Hydroelectric Power Company does not offer such terms; they are quite exceptional and are only occasionally obtainable under especially favorable circumstances. He considers that steel works on the northeast coast are in a better position to adopt the electric furnace so far as a supply of electric energy is concerned than many of their continental competitors, by reason of the huge output of the supply companies and their policy and enterprise in utilizing the waste gases from blast furnaces and coke ovens.

#### The Physical Society's Exhibition.

The Physical Society's sixth annual exhibition was held at

the Imperial College of Science and Technology on Dec. 20. The exhibits included almost all the newest and best forms of scientific apparatus from some 35 of the leading firms in Britain; and there was a very good attendance. Even to name all the various apparatus well deserving of commendation would extend this notice to an inordinate length, and only those more directly of interest to engineers will be mentioned. The Cambridge Scientific Instrument Company showed Peake's patent compensating leads for thermo-couples, the Brearley recalcence curve tracer, and a new "Bi-meter"  $\text{CO}_2$  recorder, which consists of two meters with an intermediate absorption chamber; Messrs. A. C. Cossor had Rose's high-vacuum oil pump, suited to the exhaustion of lamps; Messrs. Crompton & Company pyrometers, thermo-couples and electrical instruments; Messrs. Elliott Brothers, micrometer gages and the Wimperis accelerometer; Messrs. Everett, Edgcumbe & Company photometers and the Trotter combined accelerometer and gradient measurer; Messrs. Evershed & Vignoles, the Digby & Biggs "Dionic water tester" by which, from the conductivity, the degree of hardness, amount of condenser leakage, priming and sewage pollution may be rapidly determined; the Foster Instrument Company, a fixed-focus radiation pyrometer giving direct-temperature readings without focussing or other adjustment, and a base metal thermo-couple of high e.m.f. and low resistance; Messrs. Gallenkamp & Company, an electric furnace designed for experiments in recalcence and radiant heat; Messrs. J. J. Griffin & Sons, improved electric furnaces; the India Rubber, Gutta Percha & Telegraph Works Company, the Bleeck-Love primary battery—a new two-fluid cell giving 2.7 volts with great current capacity and constancy; Messrs. Muirhead & Company, an electromagnetic rectifier which utilizes all the wave and has an efficiency of about 90 per cent; Mr. R. W. Paul, new patterns of thermo-electric and resistance pyrometers and a thermo-electric indicator which can be read from a distance of 20 ft. to 30 ft.; Messrs. Reynolds & Branson, a lamp for testing color-vision; Messrs. Townson & Mercer, the Wanner optical pyrometer, and Somerville's  $\text{CS}_2$  and  $\text{H}_2\text{S}$  apparatus for gas testing. The optical apparatus shown by Messrs. R. & J. Beck, Messrs. J. H. Dallmeyer & Company, Messrs. A. Hilger, Messrs. Newton & Company and Messrs. Carl Zeiss was of a very high standard.

#### New Process of Extraction of Copper and Zinc from Complex Ores.

Dr. W. HOMMEL and Mr. H. T. DURANT subject a mixture of roasted ore and water to the action of cooled sulphurous acid gas, obtained from an ore-roasting furnace. The sulphite solution thus formed is collected and submitted to the action of hot gases containing sulphurous acid from the roasting furnace, the excess of sulphurous acid being thereby driven off to enrich the escaping furnace gases, and the copper or zinc is precipitated as monosulphite. The process has been patented.

#### A New Scotch Iron Ore Field.

The iron ore recently discovered in the Island of Raasay is of rather low grade, only about 30 per cent, but is quite fit for making foundry iron. The island has an area of 28 sq. miles and there is an excellent site for the construction of a harbor. After the field has been thoroughly proved, Messrs. William Baird & Company propose to erect blast furnaces to smelt the ore on the island and to take the necessary coal from Clyde ports by steamers, which will return laden with pig iron.

#### Corrosion of Copper and Iron Alloys by Aerated Salt Water

A piece of "delta" metal corroded by sea water was found by Mr. W. P. Jorissen to contain 41.1 per cent of cuprous oxide, no cupric oxide and no zinc. Cuprous oxide was also formed by the action on copper of sea water or sodium chloride solution of similar strength for two years. If copper is partly immersed in distilled water cuprous oxide is produced on the parts submerged and cupric oxide on the surface exposed to the air.

Samples of iron, some corroded underground and some by

sea water, contained both ferrous and ferric oxides and from 14 per cent to 45 per cent of substances insoluble in hydrochloric acid and which contained 50 per cent and upward of carbon. This, together with the fact that the specific gravity of the samples varied from 2.0 to 2.53, shows that a large proportion of the iron had been dissolved.

#### Preparation of Tantalum Compounds from West Australian Fergusonite.

A specimen of fergusonite from West Australia was found by Messrs. E. Wedekind and W. Maas to contain 54.22 per cent of  $\text{Ta}_2\text{O}_5$ , 27.73 per cent  $\text{Y}_2\text{O}_3$ , 1.8 per cent  $\text{Ce}_2\text{O}_3$ , 3.2 per cent  $\text{Ce}_2\text{O}_3$ , and 0.36 per cent  $\text{ThO}_2$ . After separation of the basic and acid constituents by the Krüss and Nilson method (Abtr. Chem. Soc., 1887, 706), the residue consisted essentially of tantalic, columbic and titanitic acids; 7.5 gm of the residue were carefully added to 15 gm of fused sodic carbonate in a platinum crucible, than a further addition of 4 gm of sodic carbonate and 1.6 gm of sodic nitrate was made; after cooling the fused material was extracted with 750 cc cold water, filtered and left till next day, when 6.4 gm of sodic tantalate had crystallized out. The best crystals were removed, the rest was dissolved in much hot water, and, after cooling, tantallic acid was precipitated by carbon dioxide, dried at  $100^\circ\text{C}$ ., treated with a little fuming nitric acid, evaporated to dryness, and ignited to pentoxide.

#### Market Prices.

January, 1911.

Copper has shown, on the whole, a downward tendency. Starting at £56.10.0 it had fallen by the 13th to £55.6.0, recovering sharply to £56.2.6 on the 18th, afterwards sagging and closing at £55.5.0.

Tin has made a very strong upward movement on the whole month. Opened at £179.18.0, and rose in three sharp jerks followed by equally sharp depressions to £185 on the 10th and £190.10.0 on the 18th. It has since been to £192.10.0, but closes at £190.

Lead has been steady with slight downward tendency. Opened at £13.10.0, closed at £13.7.6.

Hematite has been irregular. Opening at 66/6 it rose to 67/- by the 9th, and remained fairly steady till the 17th, afterwards dropping suddenly to 66/-, and then rising with irregular movements to 67/2½.

Alum, lump, loose.....	£5 12 6
Antimony, black sulphite powder, per ton.....	22 0 0
Borax, British refined crystal, per ton.....	16 0 0
Sal ammoniac, lump, first delivered U. K., per ton.....	42 0 0
Copper sulphate, per ton.....	19 5 0
Caustic soda, ash, 48 per cent, ordinary, per ton.....	5 10 0
Litharge, flake, per ton.....	15 10 0
Sulphur, recovered, per ton.....	5 0 0
Camphor, 1-oz. tablets .....	1 7
Shellac, standard T. N. orange spots, per cwt.....	4 4 0
Hydrochloric acid, per cwt.....	5 0
Copper ores, 10 to 25 per cent unit.....	8/10½ to 9 10½
Tin ore, 70 per cent, per ton.....	£116 to £118
Carbolic acid, liquid, 97/99 per cent, per gal.....	1 0
Creosote, ordinary good liquid, per gal.....	2
Naphtha, solvent, 90 per cent at 160° C., per gal.....	9
Mica, small slab, original cases, per pound.....	6d to 1 0
India rubber, Para fine, kept in the neighborhood of 5/4 till the 11th, afterwards falling to 4/10, thence recovered sharply, closing at.....	5 3
Platinum, per oz.....	8 0 0
Sulphate of ammonia, f.o.b. Liverpool, per ton.....	13 7 6

The most marked advance (excluding tin ore) is that of sulphate of ammonia, 10/9 per ton. Copper sulphate is down 5/-, litharge 5/- dearer, sulphur 5/- dearer, shellac up 1/-, copper ore down 6d, tin ore up £12, naphtha solvent 1d less per gallon, and india rubber 1/- cheaper.

Of the metals tin and iron have improved, copper and lead are slightly down.



## Synopsis of Metallurgical and Chemical Literature

### Iron and Steel.

**Tests of Metals by Alternate Stresses.**—A. BOUDOUARD has made tests on the influence of alternate stresses on various metals, and has summed up the results in a paper in the *Bulletin de la Société d'Encouragement* of December, 1910. A metal can be fractured, not only by applying once a breaking load, but also by submitting it to often-repeated tensions and compressions of much smaller magnitude. The resistance of metals to alternate stresses is of great importance in industrial application of steels. A metal is not perfectly elastic, and when an effort which has caused a deformation disappears the piece returns approximately to its initial form, but, nevertheless, it suffers a permanent deformation, though a very feeble one. This residual deformation is negligible compared with the elastic deformations, but very frequent repetitions can produce a profound change of the metal, and even a fracture. The first steel examined in these tests was a good commercial steel with 0.3 per cent carbon, first in its natural state, then, after having been annealed at 350° for 10 hours without access of air, and finally hardened by water quenching from 1000° C. The bar of the steel in its natural state broke after 12½ hours continued vibration, or 1,350,000 vibrations. The annealed bar broke after 4½ hours and 486,000 vibrations. The hardened bar broke after 30 minutes, and 54,000 vibrations. The rupture always took place near the spot where the steel bar was clamped, called here the clamping place. The coming rupture indicated itself in advance by a dampening of the vibrations. At the moment of breaking the free part did not fall off at once, but could only be separated from the other part by some effort. Examination of the section with the naked eye showed parallel zones with grains of different sizes, indicating the phenomenon of shearing. Following are the results of the tests with different metals:

1. **Puddled Iron.**—Iron studied in the natural state; annealed at 950° C. and cooled in air. Bar broke after 18 hours 45 minutes, or 1,995,000 vibrations. The break occurred 5 millimeters inside of clamping place, and pieces were separated with great difficulty.

2. **Very soft steel, annealed at 950°, cooled in air.** Bar broke after 11 hours 15 minutes, or 1,215,000 vibrations. Break occurred 7 millimeters inside of clamping place, and pieces were separated with great difficulty.

3. **Steel with 0.3 per cent Carbon.**—Studied first in the natural state (annealed at 900°, cooled in air), and, secondly, hardened (heated to 900° C. and quenched in 15° water). The steel in its natural state broke after 13 hours 15 minutes, or 1,431,000 vibrations. Break started several millimeters inside of clamping place. When drawn out of vice jaws, bar remained in one piece and was only separated with great difficulty. Fracture seemed to extend from the interior to the exterior. The hardened steel bar broke after 14 hours, or 1,512,000 vibrations. Fracture began 12 millimeters outside of clamping place. The sections showed results of shearing of metal particles.

4. **Steel with 0.6 per cent Carbon.**—Studied first in natural state (annealed at 850°, then at 700°, and cooled in air), and secondly, hardened (quenched from 850° in 15° water). Annealed bar broke after 6 hours, and 648,000 vibrations. Fracture showed 2-3 millimeter outside of clamping place, and the same observations as on 0.3 per cent carbon were made. The hardened bar broke after 1 hour 55 minutes, and 153,000 vibrations. Fracture started several millimeters inside of clamping place. The two sections of the bar separated without much effort. Sections showed results of shearing. Fracture is not perpendicular to longitudinal axis of bar, and found to be very irregular.

5. **Steel with 1 per cent Carbon.**—First in natural state (annealed at 850°, then at 650°, and cooled in air); second, hardened (heated to 850° C. and quenched in 15° water); third, heated to 850°, quenched in 15° oil bath, and reheated to 350° without access of air. The natural steel bar broke after 3 hours 55 minutes, and 369,000 vibrations. Fracture was produced several millimeters outside of clamping place, and same phenomena as in natural steel of 0.3 and 0.6 per cent carbon were observed. The hardened bar broke after 5 minutes, and 9000 vibrations, 7-8 millimeter outside of clamping place. This was due, however, to fracture of the bar during mounting for test. The hardened and reheated bar did not break after 26½ hours, and 2,862,000 vibrations. Tests were not continued on this bar.

The general conclusions are summed up as follows: As these tests can be repeated under same conditions several times, they can be used to find the resistance of steels to alternate stresses. Sufficiently prolonged vibrations lead to the ultimate breaking of the metal. The number of vibrations necessary for breaking varies inversely as the carbon percentage of the semi-hard and hard steels. Puddled iron and very soft steels with similar chemical composition, except for manganese, which varies between 0.050 and 0.400 per cent, present a much greater difference from point of view of length of time for vibrations to produce fracture. The puddled iron has a much greater resistance than the soft steel; 0.3 per cent carbon steel, annealed or hardened, shows no appreciable difference. In the case of hard steels hardening decreases the length of time necessary to produce break considerably. Reheating improves the metal, shown by the last case, where a 1 per cent carbon steel did not break after 26½ hours' vibrations, when reheated.

Examination of the fractured sections shows similarity. Lines of separation, along which the fracture gradually takes place, can be seen. The size of the grain varies in the different zones, and appears notably enlarged in hardened steels. Shearing lines are seen perpendicular to the edges of the fractured section. As regards the mechanical properties, the high elastic limit of puddled iron and very soft steel is pointed out, after having been subjected to vibrations leading to break. All the metals tested broke, although tested considerably below the elastic limit. Some observations were also made on the dampening of the vibrations just before metal breaks.

### Gold and Silver.

**Improved Grinding Pan Discharge.**—In the *Monthly Journal of the Chamber of Mines of Western Australia*, Mr. G. C. KLUG describes a simple expedient for obtaining a uniform discharge from grinding pans. The design is by C. C. Freeman, who introduced it at the Great Fingal mill, where it has been in successful use for over a year. It consists of a curved 3-in. pipe introduced into the side of a Wheeler 5-ft. pan opposite the feed and about 12.5 in. below the top. As the mullers revolve at 60 r.p.m. the only product which can flow out of the pipe is the —30 mesh, this being sufficiently light to be unaffected by the centrifugal force produced by the revolution of the mullers. This classifying action of centrifugal force is remarkably effective, as the overflow contains only 4 per cent to 6 per cent of coarse material, as against 10 per cent to 15 per cent which formerly was mixed with the —30 mesh when ordinary classifiers were used. The stamp battery product which goes to these Wheeler pans is crushed through a screen having seven holes per linear inch. One pan will take care of the output of five stamps. The stamp duty has been increased from 7.5 tons to 9.2 tons per day as a result of the change.

**Revolving Pressure Filter.**—A horizontal tubular revolving filter is an innovation installed at El Oro mill, Mexico, where six units are in operation, handling 1000 tons of slime per day at a cost of about 4 cents per ton. The filter is the invention of EDWIN BURT, who describes it in the Jan. 21, 1911, issue of the *Engineering and Mining Journal*. Each filter consists of a cylinder 42 in. in diameter and 40 ft. long, set in a horizontal position. The head end is closed with a casting bolted to the flanges of the shell. The discharge end is closed by a plate operated by a hydraulic cylinder and piston, working under 125 lb. pressure. The filter revolves on trunnion bearings at the head end, and on roller bearings near the discharge.

The inside of the filter is lined with 20 filter cloths, each 26 in. by 10 ft., placed with the long axis corresponding to that of the tube. This allows five cloths to the circle and 10 to the length of the tube. The cloths are held in place by curved angle irons, staggered throughout the interior in such a way that they act as a spiral guide for the discharge of the cake when it is washed. Midway in the length of each cloth is a nipple and screen inserted through the shell of the filter for the discharge of the filtered solution. As there are five cloths in the circle there are five nipple discharges in the circumference of the tube at four points in its length. Each filter cloth is a unit in itself and its discharge may be plugged independently of the others if the effluent is cloudy.

A set of five valves at the head of the filter is used in the manipulation, viz., one 4-in. slime feed, one 2-in. wash solution, one 2-in. wash water, one  $\frac{3}{4}$ -in. compressed air and one 3-in. escape for reducing pressure. The filters revolve continuously. When the doors are closed the slime feed is admitted, under a gravity head of 60 ft., until the gage shows 35 lb. pressure. The feed is then closed and compressed air passed in to bring the pressure up to 40 lb. This is maintained until the solids are caked. Wash solution is then introduced and the same procedure followed. Finally wash water is used in the same way. The pressure is then dropped to zero, the discharge door opened and the cake discharged by the action of the curved angle irons after it has fallen from the walls of the tube. An acid bath is given once a month to remove lime salts. The life of the cloths is about four months; cost of renewal, less than 0.4 cents per ton of slime filtered. Time required for complete cycle: Charging slime, 3 minutes; making cake, 23 minutes; washing, 33 minutes; discharging, 3 minutes; total, 62 minutes.

#### Copper.

**Purification of Electrolytes.**—Writing in the *Australian Mining Standard* for Dec. 28, 1910, and Jan. 4, 1911, Mr. H. SCHRODER describes work done at Lithgow, N. S. W., on the purification of electrolytes used in electrolytic copper refining.

**Crystallization Method.**—The old method used was to draw off a portion of the electrolyte each day into lead-lined evaporating tanks having a hard lead coil in the bottom, connected with exhaust steam from the engines. The solution was evaporated to sp. gr. 1.4. The evaporation amounted to about 3 in. of solution every 12 hours from a tank having a surface area of 7 ft. 6 in.  $\times$  3 ft. 6 in. While still hot the solution was syphoned into crystallizing tanks in which were suspended strips of lead on which the copper sulphate crystallized. When the solution had cooled the strips were removed and the crystals detached. The solution was then reconcentrated and recrystallized, but a third crystallization could not be advantageously effected on account of the impurity of the crystals obtained. Consequently, the impure mother liquor was passed over scrap iron contained in boxes similar to zinc boxes used in gold precipitation and the remaining copper recovered in this way. The hot mother liquor solution was delivered to the scrap-iron boxes at the rate of 2 gal. per minute and issued from the fifth box free from copper. The waste liquor, being a strong solution of iron sulphate, was discarded. Much sulphuric acid was lost by this method which might have been saved by first

saturation the solution with scrap copper. This, however, would have added to the production of bluestone, and as there was only a limited market for this the plan was not adopted.

The cement copper obtained from the scrap-iron boxes was dried and roasted and gradually worked up by charging it in small lots into the refinery with pigs of rough copper. But the elimination of arsenic was not great and eventually most of it got back into the copper anodes by way of flue dust subsequently collected and melted. Roasting the cement copper by itself was ineffective for the removal of the arsenic, for although dense white fumes came off there was less than 30 per cent arsenic driven off.

The entire method could have been improved by first saturating the solution with copper, making three batches of crystals and thereby reducing the amount of impure cement copper to be dealt with later. This latter might have been stored, mixed with flue dust or fine ore and smelted to a rich matte which could have been blown to blister and thus eliminate most of the arsenic; for while mere roasting of the cement copper is ineffectual in removing arsenic, still the reverberatory smelting and converting will remove it to a great degree, as has been shown by authorities on copper smelting.

**Experiments with Insoluble Anodes.**—These were instigated by G. H. Blakemore, and although never completed, owing to the works being closed, enough data were obtained to show the nature and results of the work. The object was to purify the electrolyte and at the same time produce the least quantity of impure copper. First a portion of the ordinary electrolyte in circulation was shunted off and circulated through a series of tanks fitted with lead anodes to reduce the copper content. The flow was regulated so that the solution leaving the last tank still contained enough copper to precipitate pure metal.

This liquor then flowed into a second set of tanks provided with insoluble anodes and circulation stopped. The current was then applied and in 8 to 12 days all the arsenic and bismuth were deposited. The cathodes were perfectly black and a large portion of the deposit fell to the bottom of the tank. The solution still carried about 2 per cent  $\text{CuSO}_4 \cdot 5\text{H}_2\text{O}$ , and was about 50 per cent stronger in sulphuric acid than the regular electrolyte. In one experiment the electrolyte contained 10 per cent  $\text{CuSO}_4 \cdot 5\text{H}_2\text{O}$ . After three days the  $\text{CuSO}_4 \cdot 5\text{H}_2\text{O}$  was reduced to 2.08 per cent and remained at that for four days more, by which time all the bismuth was removed and the arsenic reduced to a very small amount. The current density was 7 amp per square foot of cathode surface and the drop across each tank was 2.20 volts.

With a view to reducing the quantity of impure copper produced in the non-circulating tanks an experiment was run to determine how far the copper content of the ordinary electrolyte could be reduced without impurities being deposited. It was found that 7 per cent  $\text{CuSO}_4 \cdot 5\text{H}_2\text{O}$  was about the limit of safety, as below that figure the impurities would precipitate with the copper.

The author draws the following conclusions from the experiments: (1) Using a current density of 7 amp per square foot of cathode surface at 7 per cent  $\text{CuSO}_4 \cdot 5\text{H}_2\text{O}$  and up to 14 per cent free  $\text{H}_2\text{SO}_4$  (starting with 13 per cent copper sulphate and 12 per cent free acid) the cathodes darken and bismuth and arsenic begin to deposit. On two occasions, when circulation stopped during the night, it was found that at 8 per cent copper sulphate the cathodes were bright, at 7 per cent dull and at 5.5 per cent quite black. (2) Using the same current density in a non-circulating solution with insoluble anodes arsenic was removed at a regular pace. Starting with 0.54 g.  $\text{As}_2\text{O}_3$  per liter it required 12 days to lower it to 0.15 g., whereas most of the bismuth was removed in the first two days and the copper sulphate never got below 2 per cent. With 14 amp the solution was reduced to 1.7 per cent copper sulphate and the arsenic to 0.015 g. in six days.

The suggestions offered are that the copper content of the



electrolyte be reduced in lead anode tanks to the safety mark before treatment in non-circulating tanks. The whole of the impurities would then be concentrated in a small amount of copper which could be roasted with a small amount of matte and blown to blister separately, thus eliminating the arsenic to a large degree. Or the impure copper could be melted and treated as anodes in a separate tank and the cathodes, so produced, removed as long as they remained pure. The resultant solution could be concentrated and crystallized as long as pure crystals were obtainable, and the mother liquor either discarded or precipitated on scrap iron and the cement copper rich in impurities disposed of to the best advantage. In any case it would be better to discard a small quantity of copper than to return the impurities to the tank house.

**Occluded Gases in the Alloys of Copper.**—Messrs G. GUILLEMIN and DELACHANAL have made extensive studies of the occluded gases in copper alloys, and they report on the results of their investigations in the *Revue de Metallurgie* of January, 1911, as follows:

Tin, phosphor-bronzes and brass, as used in the industries and naval construction, were tested for occluded gases. For these tests quartz tubes were used, as the porcelain tubes were found to be too fragile. The opaque quartz tubes were first tested as to their impermeability to gases, both at a temperature of  $1100^{\circ}$  C. and in the cold state. In the cold they were impermeable. They were then kept under a vacuum at  $1100^{\circ}$  for ten hours, and only very slight quantities of gas were evolved, consisting of carbon dioxide and some traces of organic matter, which had entered the tubes in spite of all precautions. The error due to this is, however, so slight that it is negligible. The conclusions derived from the test are as follows: Special forgeable brasses retain important volumes of occluded gases (1 to 30 vol.) composed of hydrogen, carbon dioxide and carbon monoxide. In sound samples the volume of occluded gases (1 to 30 vol.) composed of hydrogen, carbon dioxide and carbon monoxide. In sound samples the hydrogen is accompanied by a considerable portion of carbon monoxide and a little carbon dioxide. In sound pieces the presence of occluded hydrogen, even in considerable proportions, seems to have no bad influence on the mechanical properties of brasses which can be forged. Phosphor-bronzes have very little occluded gas; this is composed entirely of carbon dioxide and hydrogen. (As in the case of cast iron and steel, phosphorus seems, therefore, to diminish the solubility of hydrogen in the metal.) The occluded gases can only be extracted by employing an increased temperature and at least a partial vacuum. The article gives a complete table showing the composition of the pieces tested, the volume of gas extracted, composition of the gases, which were  $\text{CO}_2$ , H,  $\text{CH}_4$ , CO, and some of the rare gases, like argon, etc.

**Zinc.**

### Recovery of Zinc in Liquid State Direct from Zinc Blende.

—F. THOMAS, in *Metallurgie* of Nov. 22, 1910, reports on the results of tests made for the purpose of obtaining zinc in liquid state direct from blende by electrolysis. A very fusible silicate mixture whose melting point is below the boiling point of zinc,  $920^{\circ}$ , was first thought of, and for this purpose silicates of Mn, Al, Mg, Ca, Sr, Ba, Na, K were picked out. The curve of the melting points of mixtures of  $\text{Na}_4\text{SiO}_4$  and  $\text{K}_4\text{SiO}_4$  shows a minimum ( $743^{\circ}$  C.) for 50 per cent  $\text{Na}_4\text{SiO}_4$  and 50 per cent  $\text{K}_4\text{SiO}_4$ . The further addition of 6 per cent manganese silicate reduced the melting point to  $700^{\circ}$ . The solubility of  $\text{ZnS}$  in this silicate mixture was tested up to  $1000^{\circ}$ . At  $900^{\circ}$ , the maximum temperature which can be used lest zinc is lost by evaporation, only 0.5 per cent zinc sulphide is dissolved in the silicate mixture. Electrolysis of  $\text{ZnS}$  in this silicate mixture was then tried in two furnaces of Borchers. The result of all the tests was the same, namely, a fine distribution of zinc globules throughout the mass and a separation of a very small regulus in two tests only. The probable reasons for failure are the formation of a colloidal zinc dust from the cathode and

the slow speed of solution of ZnS in the silicate mixture at temperatures below 900° C. Electrolysis under pressure was then tried in order to employ higher temperatures, and in this way 52.6 and 67 per cent zinc were recovered, although mostly in form of zinc dust. However, these tests were given up as not promising success. Further tests were then made, trying a solution of ZnS in a Na<sub>2</sub>S—FeS mixture. The intended reactions were  $\text{Fe}_2\text{O}_3 + 2 \text{Na}_2\text{SO}_4 + 11 \text{C} = 2 (\text{Fe} + \text{Na}_2\text{S}) + 11 \text{CO}$  and  $\text{Fe} + \text{Na}_2\text{S} + \text{ZnS} = (\text{FeS} + \text{Na}_2\text{S}) + \text{Zn}$ .  $\text{Fe}_2\text{O}_3$  and  $\text{Na}_2\text{SO}_4$  were reduced at 1100°. Fifty grams of the reduced mixture were mixed with 36.15 grams ZnS and heated at a constant temperature for a long time. At temperatures below 920°-960° a precipitation of zinc by iron dirt did not take place. Tests in which the fusibility was increased by adding  $\text{CaFe}_2$ , and the intention was to increase the speed of reaction by an increase of the Fe content, had a negative success.

Tests were then made to separate zinc under pressure at higher temperatures. The zinc deposited on the cover of the crucible in form of dust. In one test, which lasted 30 minutes and in which the pressure was 2-3 atmospheres, 55 per cent zinc was recovered as dust. The reaction between the Fe and S takes place near the highly heated carbon resistor, and the Zn rises as vapor upward along the carbon resistor, which is probably so hot that at its temperature the partial pressure of the zinc vapor is higher than the pressure in the electric furnace. The vapor rises, therefore, to the surface and condenses on the relatively cool walls of the iron crucible and crucible cover in form of dust.

The last tests were made with electrolysis of ZnS solution in a fused sodium sulphide-iron sulphide mixture, using Fe-anodes. The greatest percentage of metal recovered was 14.7 per cent; there was again the formation of a zinc mist or fog, which extended from the cathode. Although the results are negative, the paper is interesting.

### Assaying.

**A Portable Assay Outfit for Field Work.**—Engineers who have had occasion to prospect in isolated regions where assay offices are few and far between will be interested in a contribution to the January, 1911, *Bulletin* of the Amer. Inst. of Min. Eng. by S. K. BRADFORD. He gives full details of equipment and procedure and shows the ingenuity which can be exercised when ordinary laboratory facilities are not at hand. The equipment is packed in an ordinary 26-in. valise and suffices for about 100 assays of commercial accuracy.

**Equipment.**—Small hammer; steel mortar with stand sawed off flat to serve as an anvil; small steel pestle; blow pipe and blow-pipe cupel holder; small and large spatulas; button pliers; light 1-in. cupel mold; two 3.5-in. x 8-in. muffles; 18 "F" crucibles; six crucible covers; crucible tongs; 25-cc. burette; six 0.5-oz. parting flasks; six 0.5-oz. annealing cups; two or three beakers; hand balance, folding in a tin box; set of gram weights; one 6-in., 60-mesh screen; six small glass stirring rods; teaspoon and tablespoon; a micrometer for measuring diameter of buttons. The latter may be a short fixed microscope stand in which has been placed a scale of 0.1 mm—two or three extra object glasses being carried for safety. Chemical reagents in bottles in metal containers: 0.5 lb. KCN; 1 lb. C.P.  $\text{HNO}_3$ ; 1 lb. C.P. HCl and 2 lb. aqua ammonia. A few ounces of sheet copper; 1 oz. C.P. sheet silver; 0.5 lb. sheet lead; 2 lb. tenpenny wire nails; 5 lb. bone ash; 5 lb. sodium bicarbonate; 5 lb. litharge; 3 lb. red lead; prepared charcoal for blow-pipe use; candles and old newspapers.

It will be possible to obtain sodium bicarbonate at country stores and bone ash may, on occasion, be made in the field. Sheet lead may be made from bullets and silver can be obtained from coins of known fineness.

The assays can be fused in a blacksmith's forge, which is to be found at every mining camp, and the cupelling can be done at the same place by setting up the small muffle mentioned. The weight of the buttons obtained can be determined by meas-

uring them and referring to a previously prepared scale based on specific gravity multiplied by solid contents. This method of determining weights can be found in the *Transactions of the A. I. M. E.*, xxxi, 800 (1901).

### Recent Metallurgical Patents.

#### Gold and Silver.

**Treatment of Auriferous Sand.**—A process for treating black sand, concentrate and the like has been patented by Carl F. Siegert, of Jeppes, near Johannesburg, Transvaal. The invention consists in roasting the refractory material, or heating it to a dull red heat, and quenching it in a solution of sodium and ammonium chlorides to free the gold from oxides. The solution is then evaporated and sulphuric acid added to the comparatively dry mass. Water is subsequently added and the mass mixed. After the dilute acid has remained in contact with the ore for a suitable time it is drawn off and the mass is washed with water. It is claimed that this treatment will render the precious metal content free for recovery by amalgamation in the usual manner.

For each ton of material the inventor uses about 42 gal. of water and 10 lb. NaCl, to which is added 2 gal. of strong sal-ammoniac solution made by dissolving 2.5 lb. to 3 lb. sal-ammoniac in 100 gal. of water. This solution is used for the first treatment. For the acid wash he uses 1 lb. of sulphuric acid, more or less, as needed, and adds water after the acid has been poured on the mass. (981,696, Jan. 17, 1911.)

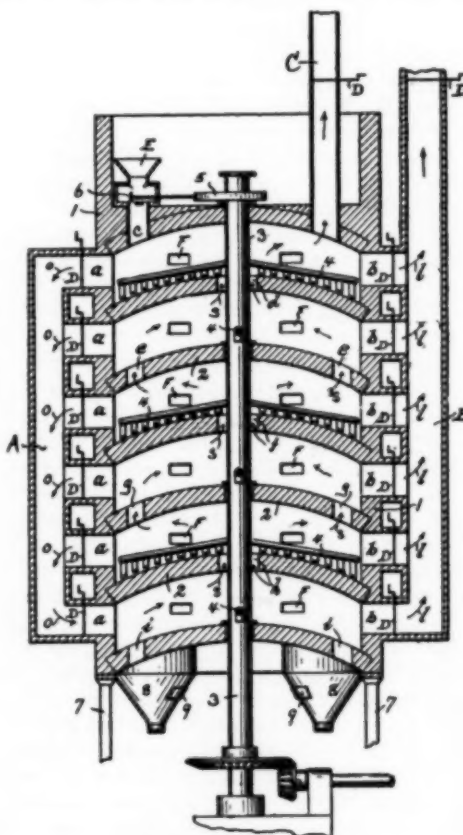
#### Roasting.

**Improved Roasting Process.**—For the purpose of improving the quality of gases produced in roasting sulphide ores for the manufacture of sulphuric acid, CHARLES W. RENWICK, of Isabella, Tenn., has patented a furnace of novel construction. In the usual roasting furnace the ore is fed at the top and passes downward over a series of hearths to a discharge beneath, and the air is admitted at the bottom and rises through the several hearths to a stack at the top. Metallic sulphides are thus roasted to oxides and as they are catalytic agents they convert  $\text{SO}_2$  into  $\text{SO}_3$  in quantities varying with the conditions. In the ordinary procedure described the inventor says that the most limited catalysis is produced. It is his object to reverse the current of gases and cause them to flow down with the ore in order to carry the high temperature found in the upper hearths down to the lower for the better roasting of the ore and in order to have the  $\text{SO}_2$  come into more frequent contact with the metallic oxides so that more  $\text{SO}_3$  will be formed.

The accompanying drawing, Fig. 1, illustrates his preferred form of apparatus. It consists of an ordinary furnace shell with two vertical flues on opposite sides, one for drawing off all or a part of the gases from any of the upper hearths and entering them into the lower, and one for drawing off all or part of the gases, depending on the amount of conversion desired, and conducting them to the place of use.

In addition to the usual construction of the type of furnace shown, the points of which are generally understood, the improved type embodies flues *A* and *B* placed at opposite sides of the furnace body and having branches *a, b*, provided with dampers *D* which communicate through openings in the furnace shell with the several hearths, so that when the damper *D* in the stack *C* is closed, and the doors *F* properly opened for the admission of air the gases can be taken from any of the upper hearths and directed into those below and passed into the flue *B*; or they can circulate up in the ordinary way after subjecting the desired quantity of  $\text{SO}_2$  to catalytic action, the damper in the stack *C* being opened for this purpose. By closing the dampered openings and all the dampers *D* except the one in the lowermost branch flue *b*, the entire draft in the furnace will be downward and through the openings *d, e, f, g, h*, and upward through the flue *B*, air being admitted at the top. All or a part of the gases can be taken from any of the

hearth and turned down through flue *A* to any of the lower hearths and carried over to flue *B*, the dampers *D* being open in the branch flues *a, b*, at the point where it is desired that the gases shall pass out of certain of the hearths and into certain of the others and thence on to flue *B* or stack *C* as desired.



ROASTING FURNACE

The inventor further claims that the reversal of the gases will increase the temperature of the gases on the lower hearths and thus break up all sulphates and give a sweet roast, in addition to increasing the conversion of  $\text{SO}_2$  to  $\text{SO}_3$ . (981,880, Jan. 17, 1911.)

#### Alloys.

**Magnesium-Zinc Alloy.**—The development of the aeroplane and airship has introduced a demand for light and strong metals or alloys. WALTER RÜBEL, of Berlin, Germany, has recently patented a new alloy of magnesium and zinc which, in his specifications, he claims has a specific gravity of 1.8; when cast, presents a resistance of 15 kilos to 25 kilos; when rolled, from 25 kilos to 35 kilos, with a coefficient of expansion of from 8 per cent to 12 per cent. Satisfactory alloys of these metals cannot be produced from more than 99 per cent nor less than 94 per cent magnesium, and less than 1 per cent nor more than 6 per cent zinc. In case of materials that are to be rolled or pressed the zinc should preferably not exceed 3 per cent. Pure zinc is not essential, as good alloys can be made if the zinc contains copper, aluminium or metals of the iron group, as iron, nickel, chromium or manganese.

The alloy is prepared by carefully melting the magnesium with the exclusion of air and adding the zinc, when a readily fusible material results, which can be cast, wrought, rolled or machined like brass. The alloy presents the fracture of fine steel. (982,022, Jan. 17, 1911.)

The stack at the Great Falls (Mont.) smelter is 506 ft. high and weighs 34,000,000 lb. About 16,000 tons of brick were used in its construction. It will withstand a wind of 120 miles per hour.



### New Methods of Obtaining Hydrogen.

In the Linde-Frank-Caro process for the separation of hydrogen from water-gas the basis of the process is the production in the first instance of water-gas. As is well known this is obtained by passing steam over red hot coke whereby the former is disassociated into free hydrogen and oxygen. The oxygen combines with the carbon of the incandescent coke to form carbon monoxide, while the hydrogen remains in a free state. In addition to these two gases, commercial water-gas also contains traces of carbonic acid and nitrogen to the extent of from 3 to 5 per cent of the total volume of the gas. Water-gas consists apart from these impurities of about equal parts of hydrogen and carbon monoxide.

There are a number of ways for the chemical production of hydrogen but by the application of fractional condensation it may also be obtained in the form of a gas by means of a very simple mechanical process. Separation of the carbon monoxide from the water-gas may be brought about by cooling the water-gas to the temperature of liquid air whereby the carbon monoxide is liquified—the conditions of liquification approximating to those of nitrogen—while as is well known much lower temperatures are necessary to liquify hydrogen.

In order to carry out this process on commercial lines, the carbonic acid in the water-gas is in the first instance extracted in some convenient manner, such for example as its absorption by caustic soda. The water-gas is then compressed to a pressure of 60 atmospheres by means of a three-stage compressor. The gas is then dried and passed into an apparatus in which it is subjected to a bath of liquid air. As the result of cooling in this bath of liquid air, the carbon monoxide liquifies and the hydrogen passes out of the apparatus in the form of a gas.

The carbon monoxide may be re-evaporated or otherwise collected as a by-product and utilized in other ways. It may be employed to drive a gas engine or it may be used as fuel under a steam boiler. By using it for power purposes the whole process becomes very economical as in this way the power necessary for the fractionation of the gas can be obtained.

The process is very inexpensive to operate and the apparatus requires very little attention.

The rights in the process for this country have been acquired by The Linde Air Products Company of Buffalo.

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In this connection a paper by Director RICHARD BLUM, of the Berlin-Anhaltische Maschinenbau Company, read before the Aero Club in Berlin is interesting, as the author discusses the different processes now available for producing hydrogen gas, with special reference to the filling of balloons. The paper is printed in full in the *Journal für Gasbeleuchtung* of Jan. 21 and 28.

The oldest methods of making hydrogen gas on a practical scale, used before the eighteenth century, were based either on the action between sulphuric acid on iron or on the passing of steam over iron filings heated to a temperature of red heat. The latter process has recently been taken up again and changed to a cyclic process by the Internationale Wasserstoff Company. They employ steel retorts filled with porous iron oxide. The retorts are placed in a furnace and are heated from the outside. When the temperature has risen to a sufficient degree water gas is passed through the retorts and the iron oxide is thereby reduced to a metallic iron. When all the iron oxide is reduced to a porous mass of iron, the passage of water gas is stopped and steam is passed through the retorts whereby hydrogen is formed and the iron is again oxidized. Then the passage of steam is stopped and water gas is again passed through the retorts and so on. In this way hydrogen is produced and the iron oxidized, and in the next stage the iron oxide is reduced again to metallic iron. It is advantageous to keep the furnaces in continuous operation. The hydrogen thus produced is very pure and is compressed in steel cylinders of 6 cu. m each to 150 or 170 atmospheres.

Numerous plants are in operation in Europe for the decomposition of water into hydrogen and oxygen by electrolysis. In an apparatus for this purpose built by the Schuckert Company the production of 1 cu. m of hydrogen and  $\frac{1}{2}$  cu. m of oxygen requires 13 hp-hours. According to the present author the process is expensive and requires a large floor space. But the hydrogen produced is very pure.

Enormous amounts of hydrogen gas are produced in the large works for the production of caustic soda and chlorine by electrolysis of common salt solutions. The hydrogen gas is set free together with the caustic soda at the cathode and the quantity of hydrogen gas produced is chemically equivalent to the quantity of the chlorine produced at the anode. The quantities are so large, compared with the demand which exists at present for hydrogen, that most of the hydrogen gas is passed unused into the air. The Griesheim-Elektron Company in Germany produces daily 20,000 cu. m of hydrogen gas of about 90 per cent to 97 per cent purity. In this case the cost of the gas is practically that of compressing and storing the hydrogen gas. Special railway cars are built in Germany for the transportation of 500 cylinders containing 2750 cu. m of hydrogen gas. Much of the gas used in balloon trips in Germany is procured in this way.

Another method for producing hydrogen gas is by means of certain metals or compounds which in contact with water or alkaline solutions develop hydrogen.<sup>1</sup> Such materials are aluminium, silicon, and calcium hydride.

The production of hydrogen from aluminium and caustic soda solutions was used in practice by the Russian army in their war with Japan. This method is expensive.

More recently the Griesheim-Elektron Company has placed on the market a so-called "activated" aluminium (aktiviertes aluminium), 1.25 kg of which yield 1 cu. m of hydrogen. In this case the aluminium is stirred with water in the gas producer. According to statements of the manufacturing company the cost of producing hydrogen by this method is about 12.5 cents per cubic meter.

The Siemens & Schuckert Company has worked out a process for the production of hydrogen from the reaction between silicon and caustic soda solution. Formerly steam was employed, but now the heat set free during the reaction is utilized for maintaining the proper conditions. The evolution of hydrogen gas takes place when a 25 per cent solution of caustic soda acts on silicon introduced in small quantities. The capacity of a transportable plant (placed on wagons and drawn by two horses) is 60 cu. m to 120 cu. m per hour, while stationary plants are built for capacities up to 300 cu. m per hour. The Italian army has such plants in operation. The process is elegant, but the cost is about 18.75 cents per cubic meter.

For the filling of balloons it is of great importance to have the hydrogen as cheap as possible, say, at a price of 2 cents to 4 cents per cubic meter. This is possible by some newer processes. The first one of these mentioned by Mr. Blum is that of Linde, Frank & Caro for producing hydrogen from water gas with the aid of fractional condensation. This process is described above. In Mr. Blum's paper it is mentioned that under conditions as that exist in Germany the cost of the production of hydrogen gas by this method ranges from 2 cents to 3.25 cents per cubic meter, according to the purity of the gas and the size of the plant.

Another process worked out by the Carbonium Company in Germany employs acetylene gas which is compressed to two atmospheres and exploded by an electric spark. The acetylene thereby dissociates into the elements carbon and hydrogen. The carbon deposits in form of lamp-black. The hydrogen is passed through large washers and stored. The German plant of this company was destroyed last summer by an explosion. A condition for getting the hydrogen cheaply by this method is that there is a market for the lamp-black.

<sup>1</sup>The production of hydrogen gas from hydrone (made by the Roessler & Haaslaicher Chemical Company) belongs to the same class.

Another method of the Berlin-Anhaltische Maschinenbau Company employs the system of Rincker & Wolter. The starting materials are oils, petroleum, benzine or any heavy hydrocarbons. The process has been first developed as a substitute for illuminating gas plants using bituminous coal. These plants can be so equipped that they produce in general illuminating gas and gas for heating and cooking, and in exceptional cases hydrogen gas.

Plants of this kind are also instilled on railway cars to ship them to any place where the hydrogen may be wanted. There are two gas producers lined with fire-brick. They are filled to three-quarters with coke. By means of a steam turbine and a blower air is blown into the gas producers and the coke blown red hot. The turbo blower is then stopped by means of an oil pump and oil is sprayed into a generator. The oil is gasefied at once with deposition of lamp-black, while the hydrogen gas is passed through a washer and cooler and dryer, etc. The capacity of such plants is about 100 cu. m or more per hour. The time required for putting the plant in operation is two or three hours; it is then able to produce hydrogen gas continuously.

Hydrogen gas plants are furnished from Germany to Austria, Italy, and Russia, so that there seems to be a market for hydrogen in Europe. In the past it has been restricted in this country.

next pass through an absorber, in which they are brought in contact with lye solution. This solution has the property of absorbing carbon dioxide, so that the flue gases emerge from

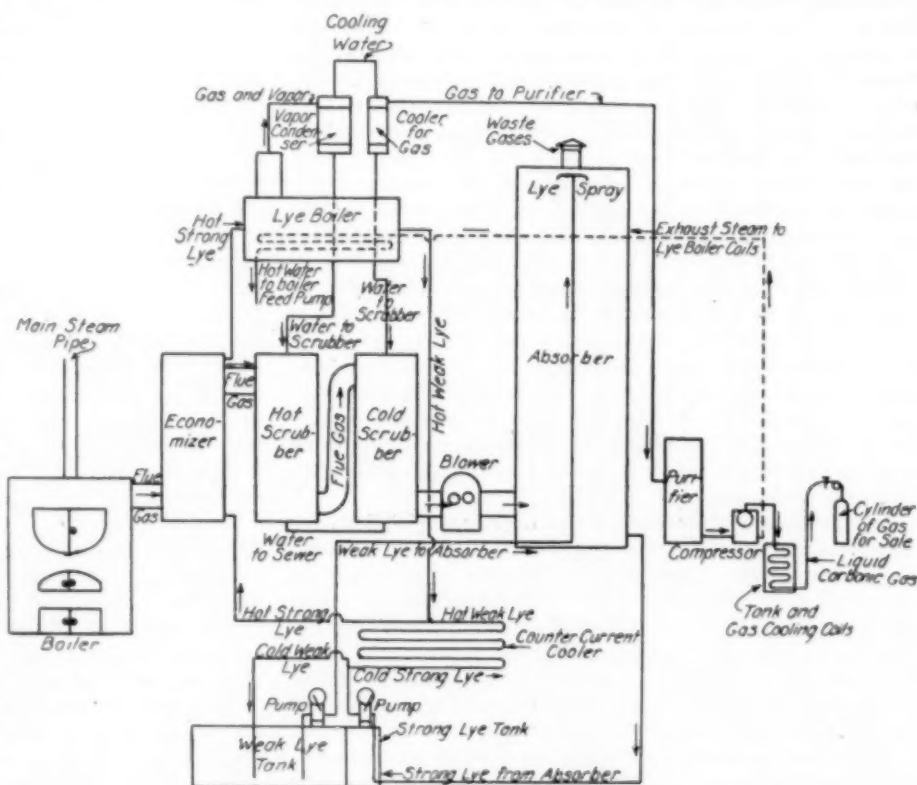


DIAGRAM OF CARBON DIOXIDE MANUFACTURING PLANT.

### A Problem in Saving Heat in the Manufacture of Carbon Dioxide.

BY STERLING H. BUNNELL.

A very large business is done in the manufacture of carbonic gas for use by bottlers and others. This gas, pure carbon dioxide, is obtained from the combustion of coke, washed to take out sulphur and other impurities, separated from the other flue gases, dried, compressed to the liquefying pressure and bottled up in steel drums or cylinders for sale. The problems which must be solved in connection with making an economical plant for the manufacture of liquid carbonic gas entail almost every form of steam plant engineering.

The first operation in manufacturing is to burn the foundry coke under a steam boiler. A deep fire is carried and a forced draft used, so that the flue gases may have the highest possible proportion of carbon dioxide. With a properly designed furnace and good firing the proportion of carbon dioxide will run above 14 per cent and may at times reach above 16 per cent. The flue gases leaving the boiler setting are at a high temperature; this heat must all be disposed of before the gas is in condition for absorption. Here is the first point at which waste heat is to be saved. Accordingly, the flue gases are passed through an economizer, the duty of which will be explained later. The gases leave the economizer at a temperature below the boiling point of water.

The gases next pass through two scrubbers in succession, in which they are washed by showers of water to get rid of the cinders and whatever sulphur may be present. In this operation they are cooled to the temperature of the water supplied and this portion of the heat is necessarily wasted. The gases

the absorber with a content of carbon dioxide of 10 per cent.

The lye solution from the absorbing tower is allowed to flow into a tank. This part of the solution is called the "strong" lye, because of its containing a considerable quantity of carbon dioxide. The next step is to pump the strong lye into a lye boiler, where it is heated above the boiling point, a portion of the water is evaporated, and a large part of the absorbed carbon dioxide driven off by the heat. Here again is a point at which heat must be applied continuously to a solution, and as the temperature required is little if any higher than 212°, exhaust steam from the power portion of the plant is used for the purpose of supplying heat. The lye solution returns from the lye boiler to a second tank and as its contents of carbon dioxide have been much reduced this solution is now known as the "weak" lye. In order to permit the weak lye to be again used for absorbing a fresh supply of carbon dioxide it must be cooled from its boiling temperature to perhaps 120°.

Observing that the weak lye from the absorber is cooled and must be heated, and the strong lye from the lye boiler is hot and must be cooled, the natural suggestion is to exchange the heat between the two solutions through a countercurrent cooler. This is of the double-pipe type, the hot weak lye from the lye boiler flowing through the inner pipes and the cold strong lye from the strong lye tank flowing through the outer pipes in the opposite direction. In this way an exchange of heat takes place which returns the weak lye at a temperature nearly as low as the temperature of the strong lye on its way from the tank, and delivers the strong lye at a temperature nearly as high as that of the weak lye coming from the boiler.

The strong lye leaves the countercurrent cooler at a temperature which is still considerably below the boiling point, and here comes in the opportunity to use the waste heat of the flue gases from the steam boiler by passing the strong lye through the economizer before mentioned. The strong lye leaves the economizer at a temperature near to boiling, and in passing



the economizer abstracts a large part of the waste heat in the flue gases.

All the waste heat in flue gases has now been accounted for. The bulk of the heat caused by the combustion of the coke has gone into the steam boiler and been devoted to the work of evaporation. This affords a good supply of steam available for doing the work of the plant. Most of the work required is for the compressors, which take the gas after it has been driven off the hot strong lye in the lye boiler and successively passed through coolers and dryers. Another portion of the steam is required for operating the pumps necessary for circulating the lye and for pumping cooling water about the plant. The exhaust steam from compressor and pumping engine still carries the latent heat which it received in the steam boiler. This large supply of heat is devoted to heating the lye in the lye boiler, evaporating a portion of the water and driving off the gas.

The skill required in designing and operating the entire plant is obviously devoted to arranging the closest possible balance between the heat transfers in all parts of the plant. The capacity of the plant—that is, the amount of carbonic gas that can be put into cylinders for sale—depends first upon the amount which can be driven off the lye in the lye boiler; a step further back, upon the proportion of the gas which can be recovered from the flue gases in the absorber; a step back of this, upon the proportion of the coke which can be burned into carbon dioxide rather than carbon monoxide, and this again depends upon the efficiency of the firing, the thickness of the bed of coke and the regulation of the draft, so that no free air shall pass through the fire. If the absorption of the gas is efficient the amount of coke burned, and therefore the grate surface, can be decreased. This will decrease the amount of steam made by the boiler, the available power of the plant and the amount of heat which can be recovered from the exhaust steam. Evidently, with a very efficient absorption it might be possible to produce enough gas from so little coke that there would not be steam enough to operate the plant, and in that case every possible saving in steam consumption by engines and all waste by radiation by steam pipes should be cut down as far as possible.

The operation of a complete modern plant for manufacturing carbonic gas would be a profitable study for any steam engineer interested in getting the most out of his steam plant. It is safe to say that very few steam plants of sizes in the neighborhood of 100 hp are constructed with any such degree of effort to reduce the usual losses to the minimum as is observed in the design and operation of the modern carbonic acid gas plant here described.

*Griscom-Spencer Company,  
New York City.*

The ball-tube mill is being advocated by cyanide metallurgists as a machine that probably will give good results in preliminary crushing and grinding. These machines have given good results in the cement industry, and it is suggested that they could replace rolls in certain metallurgical operations. Either rolls or ball-tube mills would find preference with American engineers, who regard the use of the stamp as a coarse crusher as inferior to the machines mentioned.

The American Smelting & Refining Company has issued a statement showing a comparison of the condensed balance sheets for Oct. 31, 1910, and April 30, 1910. A similar statement is issued for the American Smelters Securities Company. Both statements show an improvement in earnings as compared with the previous year, and an increase in the quantity of metal held in process. President Guggenheim states that the April, 1911, statement will be more than usually extensive and comprehensive. He further adds that the supply of ore coming to the company's plants is showing a marked improvement, and that they have been encouraged to increase expenditures for improvement and new construction.

### Centrifugal Clarifier and Filter

An article published elsewhere in this issue (page 147) contains interesting advance information on a centrifugal machine just perfected after many years' work by the DeLaval Separator Company for the continuous separation of liquids and solids. A smaller machine of the same company, which has solved a special problem of the separation of solids from liquids—namely, the separation of a small amount of sediment, etc., from a solution—has already stood the test of practice in the clarification and filtration of varnishes and allied products, oils, pharmaceuticals, patent medicines, fruit syrups and extracts, in fact, of many liquids containing either a sediment deposit or the finest flock or both. But as this centrifugal machine will undoubtedly prove useful beyond its present

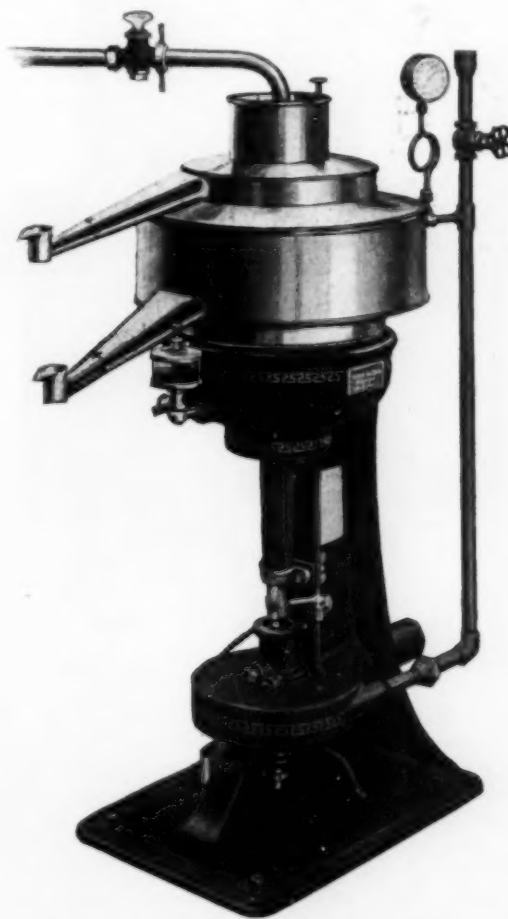


FIG. 1.—STEAM-TURBINE CLARIFIER AND FILTER.

already large sphere of work the following notes on its construction and operation should prove welcome:

An outside view of this machine is given in Fig. 1, which shows the steam-turbine style for direct steam connection. The largest size of this type has a height of 46 in. and requires a floor space of 18 in. by 24 in.—which indicates a big saving in floor space over other separating methods. In works where shafting is already installed the machine may be used with belt drive, while the smaller sizes of the machine are entirely practicable for running by hand.

Fig. 2 is a section of the clarifier and filter bowl. It contains the two compartments where the work is done, namely, the lower clarifying compartment and the upper filtering chamber. The clarifying compartment consists of a cylinder filled with a series of conical-shaped, pressed-steel disks, one above the other, about 1/32 in. apart and held rigidly in position by a central feeding device. The filtering chamber or upper section is also cylindrical in shape, but of less diameter.

The liquid first enters the bowl at the point *A* through the feeding device in the center and drops to the bottom of the clarifying compartment *B*, where much sediment is retained. It is then forced outward by the great centrifugal force through openings at the bottom to the periphery of the compartment *C*,

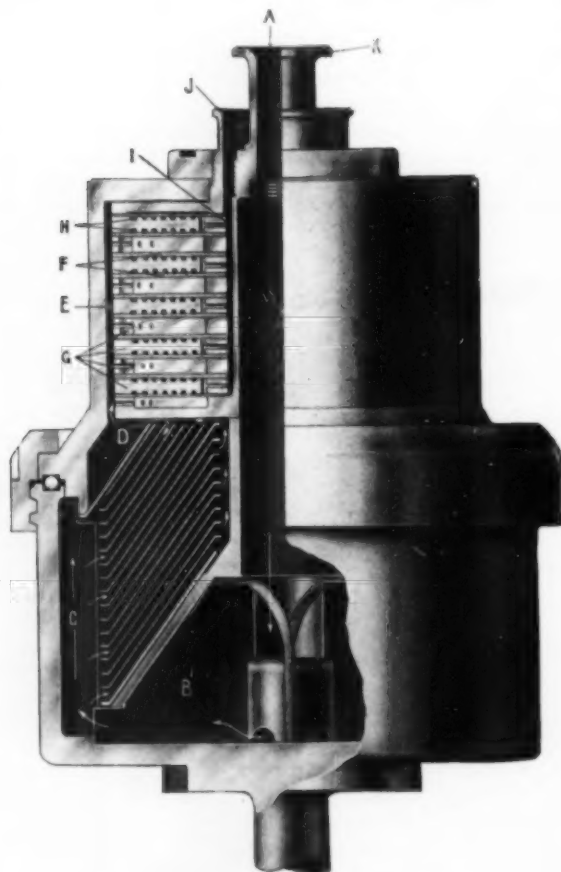


FIG. 2.—BOWL OF CLARIFIER AND FILTER.

where the bulk of heavier sediment is deposited. The liquid is next drawn to the center between the several disks. The purpose of these conical disks is to divide the liquid into thin layers, thus the friction on the disks above and below causes it to revolve rapidly and the heavier part, or balance of the sediment, is gradually forced to the under side of the disk above, whence down and out to the pocket *C*, while the thoroughly clarified liquid passes to the center and is forced upward and out into the second or filtering chamber at *D*. When the liquid enters this compartment fully 99 per cent of the sediment and foreign matter has been removed, thus leaving a small though very important work for the filter to accomplish.

The type used is what is termed the multiple filter. The filter chamber is filled with a series of horizontal corrugated plates and perforated retaining rings, between which are placed one or more thicknesses of filter material, paper, felt, cloth or whatever may be best adapted to the work in hand. The liquid is forced from compartment *D* upward into the compartment *E*, which is a narrow space between the periphery of the filtering chamber and the filtering disks. This allows a channel for the liquid to pass up, from which it is delivered into the several compartments *F*, where it is forced through the filtering material *G* into compartment *H*. Here it is again forced to the center and upward through channel *I* to the discharge point *J* and into the large cover from which it flows into the receptacle awaiting it.

The functions of the clarifying and filtering compartments supplement each other very thoroughly. The liquid is so

thoroughly clarified by the disk system in the lower clarifying compartment that it leaves but a small amount of work for the filter to perform. Then, too, the filters being placed horizontally are substantially self-cleaning, for as the sediment or flock forms in any volume on the filters the centrifugal force throws it off and it is deposited at the outside or periphery of the compartments. Thus the filter material will do its work very much longer than is possible otherwise and continue to deliver a product of the required fineness.

This machine is built by the DeLaval Separator Company, 165 Broadway, New York City. The manufacturers state that with due care this clarifier and filter will last a lifetime. The average operator finds no trouble whatever in adjusting or cleaning any of the machine's parts and keeping it ever ready to perform its work. The chief advantages claimed for this machine are as follows:

The DeLaval clarifier and filter will take any product to which it is adapted immediately after compounding and deliver continuously (except for occasional cleaning of the machine) a finished article ready for the market. It thus greatly reduces the capital necessarily tied up in settling tanks with other methods.

On an average it will make a saving of 90 per cent in filter paper and cloths and their subsequent washing and cleaning over that required by other processes, also a saving of liquid absorbed by the filter paper and that lost by evaporation or dripping with other methods, a saving of liquid, labor and the

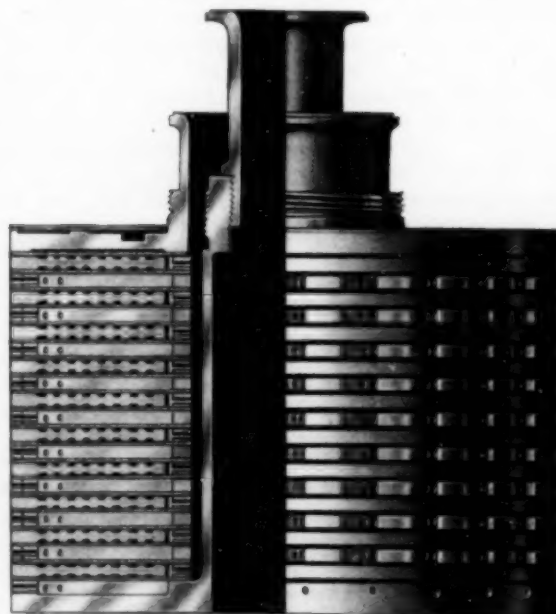


FIG. 3.—FILTERING CHAMBER.

losses that sometimes occur from punctures in paper and a big saving in floor space.

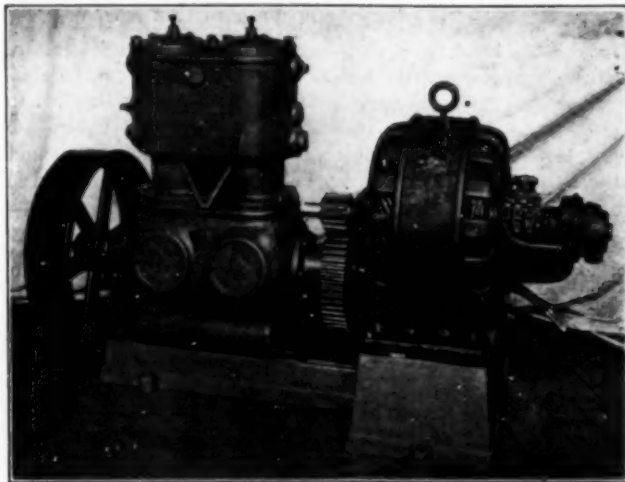
Where it is desirable to aerate the products to be handled the DeLaval machine can be made to automatically aerate the liquids as they pass through it and thus do away entirely with the necessity of air pumps or other equipment and labor necessary for aerating purposes with other filtering processes.

**Transvaal Gold Production.**—The number of companies reporting to the Transvaal Chamber of Mines in November, 1910, was 69. The total tonnage milled during that period was 1,843,941 tons of ore; 9746 stamps were in operation with an average duty of 7.32 tons per 24 hours. Tube mills in commission numbered 194. The yield for the month was 642,591 fine ounces of gold. Compared with the preceding month the only change of importance was in the increased number of tube mills in operation.



### Sulphuric Acid Handled by Compressed Air.

In the manufacture of certain chemicals considerable danger, difficulty and expense are caused by the rapid deterioration of piping and receptacles with which the acids and fumes come in



MOTOR-DRIVEN AIR COMPRESSOR.

contact. The difficulties which the destructiveness of sulphuric acid, for instance, causes in pumping are well known.

The most satisfactory method of pumping destructive acids is that used by the Nichols Syndicate in their new plant at Bay Point, Cal. At one stage of the process of manufacture the acid is accumulated in a cylindrical tank which is provided with suitable inlet and outlet valves. When it is desired to pump the acid into other retainers the inlet valve is closed and air at from 60 lb. to 100 lb. pressure is admitted, which forces the liquid through the outlet valve to other parts of the plant. The flow of the acid is controlled by suitable valves in the piping system.

To meet the increasing demand for compressed-air apparatus for such work the United Iron Works, of Oakland, Cal., have brought out the motor-compressor shown in the above illustration. The compressor is a 6 x 6 Gardner Rix Duplex; speed, 2000 r.p.m.; pressure, 60 lb. to 100 lb.; capacity, 40 cu. ft. of air per minute. The compressor is driven by a Westinghouse type HF three-phase, 60-cycle, 550-volt motor.

This type of motor is admirably suited to this work. The secondary is phase wound and the three phases are connected to the three collector rings shown. In starting, the controller gradually cuts out the starting resistance which is connected to the secondary through the collector rings. This motor is so designed that an unusually high starting torque is developed, with comparatively low starting current.

The European Spelter Convention has announced that present prices will be maintained through March, but that on April 1, 1911, there will be an advance of 0.25 mark per kilogram. A graduated price list, with concessions for wholesale purchasers, will be adopted.

### Determining the Accuracy of Indicator Springs

By E. K. UEBE.

The indicator is unquestionably the most necessary instrument for the operating engineer in a steam plant.

The indicator diagram is the only means by which to ascertain the action of the steam in the engine cylinder, to determine the efficiency of an engine and to detect internal wastes. It is, therefore, of vital importance that the indicator can be relied upon to be absolutely accurate in its performance, which especially applies to the piston spring, the most essential part of the indicator.

Unless the exact tension of the spring is known errors will result from its use and the determinations made with the indicator will be of little or no value. An instrument for testing indicator springs has recently been placed on the market by the Schaeffer & Budenberg Manufacturing Company, of Brooklyn, N. Y.

The instrument, a general view of which is given in Fig. 1, consists essentially of a closed vessel made of cast iron capable of resisting internal steam pressures up to 200 lb. per square inch. The steam pressure in the vessel is measured by a gage of special construction consisting of a piston of  $\frac{1}{2}$  sq. in. area and which is free to move in an inclosing cylinder.

The lower portion of the piston is pointed and rests in a yoke which is suspended on the knife-edge of a pair of scales mounted on top of the closed vessel. If the scales are previously balanced, before admitting steam into the vessel, it is evident that the reading of the scales will give the pressure

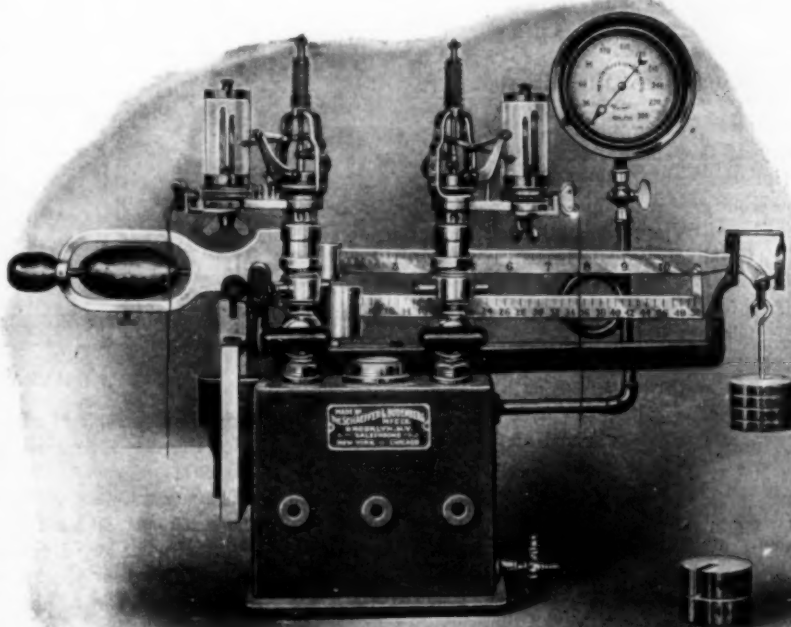


FIG. 1.—APPARATUS FOR DETERMINING THE ACCURACY OF INDICATOR SPRINGS.

acting on each element of the vessel equal in area to that of the piston. The scales are graduated to fiftieths of a pound, which permits of very close readings.

In testing indicator springs steam is turned on into the vessel to gradually warm it and allowed to blow through the connections and indicator cocks to remove any particles of dust or grit which may have accumulated there. Steam is then shut off again and the indicator screwed in its place, putting paper on the indicator drum, on which paper two vertical lines are ruled as 1-2 and 3-4 (see Fig. 2).

The indicator pencil is pressed against the drum and a horizontal line *Q Q* is drawn at the point thus marked. Then

the poise of the scale is set, say, 5 lb. and steam is admitted into the vessel, gradually rising in pressure until the scale floats. When the pressure is exact the line *RR* is drawn, and so on for successive pressures of 5 lb., until the limit of the scale is reached. During this operation the motion of the indicator pencil is continuously kept upward. If at any moment the pressure rises too high it is lowered by manipulating the valves below the required amount and then gradually raised to the desired point.

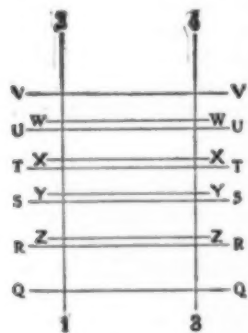


FIG. 2.—RECORDS OF TESTS.

When the indicator pencil has reached its highest position the steam pressure is allowed to fall gradually and a series of similar lines, as *WW*, *XX*, *YY* and *ZZ*, drawn during the descent of the piston. The motion being continuously downward. The distance between any two of these lines, as *ZR* or *YS*, indicates the loss due to friction of the indicator.

The special advantage of this machine, besides its absolute accuracy, is that the indicator springs are tested in the individual indicator in which they are used and under actual steam pressure which is as near to the actual working conditions as possible.

The indicator spring-testing apparatus serves also for testing pressure gages which can be connected to the three openings in the front side of the vessel provided for that purpose, and the readings are verified by the indications on the scales and the test gage shown in the illustration.

#### A New Recording Gas Meter.

A new meter for indicating, integrating or graphically recording the quantity of flow of gases at any pressure and at any temperature (independent of fluctuations of pressure or temperature) has recently been designed by Prof. Carl C. Thomas, of the University of Wisconsin, and has been placed on the market by the Cutler-Hammer Manufacturing Company, of Milwaukee, Wis.

The operation of the meter is based on a simple and interesting principle which will be understood from the diagram, Fig. 1, which shows a passage *A* through which gas is flowing in the direction of the arrow. Within the passage is an electric heater *B* made of a suitable resistor, and two electric resistance thermometers *E* are provided at the entrance and exit of the passage way. Now suppose that the voltage at the heater terminals remains constant and that the resistance of the heater does not change, then the heater will give off heat to the gas at a constant rate.

If now the rate of flow of the gas remains uniform and constant, the gas is raised by the heater in temperature by some fixed number of degrees, shown by the difference of the readings of the two thermometers *E* and this temperature difference will remain constant so long as the conditions do not change. On the other hand, if the rate of flow of gas increases (while the liberation of heat from the heater remains constant) the larger quantity of gas passing in a given time will be raised less than before in temperature; that is, the temperature difference measured by the two thermometers *E* will be smaller than before. Conversely, if the rate of flow of gas decreases, the temperature difference between the two thermometers *E* will increase. In brief, as long as the rate of liberation of heat from the electric heater is kept constant the temperature difference between the two thermometers *E* varies inversely as the rate of flow of gas, and can, therefore, be used as a measure of the rate of flow of gas.

The inherent weakness of this method, as just stated, for commercial service is that it requires the maintenance of an

exactly constant rate of liberation of heat from the electric heater. This is a condition difficult to fulfill in commercial service. Therefore, for commercial-service meters the principle of operation has been modified. The change is as simple as it is ingenious.

Not the rate of heat liberation is maintained constant and the temperature difference between the two thermometers *E* measured, but the latter temperature difference is maintained constant artificially by varying the rate of heat liberation from the electric heater—that is, by varying the electrical power supplied to the heater. Hence the temperature difference between the two thermometers *E* remains constant and the varying input of electric power in watts supplied to the heater is measured and serves as a measure of the rate of flow of gas. The watts are measured by recording wattmeter or integrating wattmeter.

The two resistance thermometers are in the form of ruggedly made screens of resistor wire, so disposed as to come in contact with all the gas passing through the meter. They form two arms of a Wheatstone bridge, which by means of the deflection of the galvanometer needle causes the controlling mechanism to cut in or out enough electrical energy so as to keep the temperature difference between the two resistor wires *E* constant. There is no necessity to go further into details as the principle is clearly one of electrical resistance thermometry, which has reached such a high state of excellence in recent years.

The accuracy of these meters is not affected by changes in pressure of the gas or air, since the unit of measurement is that of weight rather than that of volume—that is, the meter takes cognizance of the specific gravity. Also variation of temperature of the incoming gas does not effect the accuracy of the meter because it is a difference of temperature, rather than a fixed temperature, upon which the measurement depends. The meter can, therefore, be used for gas or air at either high

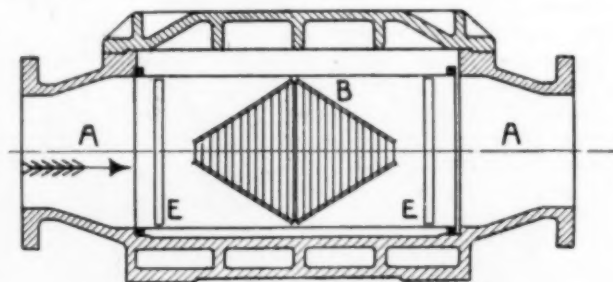


FIG. 1.—DIAGRAM OF GAS METER PRINCIPLE.

or low pressure, and at either high or low temperature, provided the materials used in construction are suited to the conditions.

The meter is especially well adapted for measuring the discharge of gas or air from compressors, blowers, etc., because the small and rapid periodic fluctuations of pressure, due to either suction or discharge conditions, do not interfere with the steady action of the thermometers. The time lag of the latter is sufficient to smooth out the curve of watts input or of temperature variation, as the case may be, and true average results are thus indicated. The fact that the meter is capable of accurately measuring gas under varying pressures and temperatures makes it especially valuable for measuring natural gas, as well as compressed air.

The electrical power required to operate the meter is approximately 1 kw per 75,000 cu. ft. per hour at atmospheric pressure.

The Thomas meter is essentially applicable to the measurement of gas or air which is either dry, saturated or superheated. If water is carried along mechanically as a fog or mist (as may be the case in blast-furnace gases coming from the cleaning apparatus) the evaporation of this by the meter itself, of course, interferes with making accurate measurements. But such fog or mist can readily be transformed into dry vapor by the introduction of heat from a small steam radiator consisting of a coil of pipe in the entrance to the



meter. Steam is usually available, and can be introduced at about atmospheric pressure or above, as may be most convenient. A very small amount of steam suffices for this purpose.

Let us now briefly consider the theory of the meter.

Let  $G$  = cubic feet of gas per hour,

$E$  = energy in kilowatts,

then B.t.u. per hour =  $3412 E$ .

$T$  = temperature difference, deg. Fahr.,

$S$  = specific heat per cubic foot.

The  $GST$  = heat energy equivalent to  $E$ , or

$\frac{GST}{E} = \frac{3412}{S} = \text{constant } K$  which depends upon the specific heat of the gas. For a gas of constant specific gravity the value  $GT \div E$  is, therefore, a constant.

That this is correct is brought out by experimental tests, the

The cross-section paper on the recording watt-meter is ruled so that  $86,250 E$  is read directly, instead of the watts  $E$ . The record is thus read directly in cubic feet of gas.

The regular records of chemical analysis of the gas should be referred to from time to time in order to ascertain what percentage variation, if any, takes place in specific heat. It appears that the elements which vary during the operation of a gas plant are not those whose variation produces serious variation in specific heat. The variation that does take place is apparently well within the limits of accuracy practicable, or generally considered necessary in the operation of gas plants. By making frequent chemical analyses the error can be reduced so as to be quite negligible.

The conditions during the air tests of Fig. 2 were as follows: Barometer, 29.75; pressure, 6.5 in. water; average temperature of air as measured in the wet meter,  $60^\circ$  Fahr.

From the air curve obtained under these conditions (Fig. 2).  $K = 188,000$ , and  $S = 3412 \div 188,000 = 0.0181$ .

Reducing this to standard conditions of 32 and 29.9 in. mercury,  $S = 0.0191$ . This is to be compared with the accepted specific heat of air under these conditions, or 0.0192 b.t.u. per cubic foot.

This is probably excellent evidence of the accuracy of these tests and of the meter.

Aside from its accuracy, based on the correct application of exact specific principles, the Thomas gasometer has some obvious great practical advantages. Compared with a wet gas meter, such as is commonly used for measuring the flow of gas, the Thomas meter is very much smaller for the same

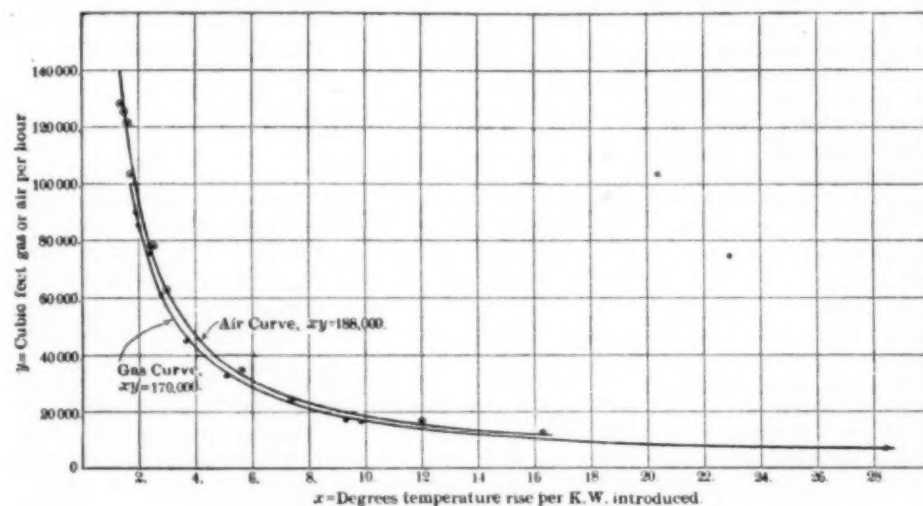


FIG. 2—RESULTS OF TESTS OF GAS METER.

results of which are given in Fig. 2. The abscissas give values of  $T \div E$  or degrees of temperature rise per kilowatt of electrical input. The ordinates give values of  $G$  or cubic feet of gas per hour. Since according to the above formula  $GT \div E$  should be a constant, the curves of Fig. 2 should be rectangular hyperboas. This is indeed the fact.

Since the temperature difference  $T$  is kept constant it follows that  $\frac{K}{T}$  is constant. Let  $\frac{K}{T} = C$ . Then  $G \frac{KE}{T} = CE$ .

It can now be shown that if the specific heat of gas made under given conditions be calculated from the customary chemical analysis and the specific heat of the constituents, then this specific heat may be used for determining the constant  $C$ . From the gas curve (Fig. 2), which was made with illuminating gas at an average temperature of  $59^\circ$  Fahr. and under an average absolute pressure of 6 in. water and 29.8 in. mercury.

$$K = 170,000 = \frac{3412}{S}$$

Therefore, for the condition of the gas when the tests were made the specific heat per cubic foot,  $S = \frac{3412}{170,000} = 0.0201$ .

If this be reduced to standard conditions of  $32^\circ$  Fahr. and 29.9" mercury then  $S = 0.021$ , which is to be compared with the calculated specific heat, giving  $S = 0.0211$ .

If the standard conditions are taken as  $62^\circ$  Fahr. and 29.9" mercury the specific heat becomes 0.0198 and the constant becomes  $K = \frac{3412}{0.0198} = 172,500$ , nearly.

If the temperature difference is kept constant at  $2^\circ$  Fahr.,

then  $\frac{K}{T} = \frac{172,500}{2} = 86,250 = C$ , or  $G = 86,250 E$ .

capacity. Moreover, its switchboard may be placed at a distance from the pipes through which the gas flows, if this is desired, thus permitting the recording mechanism to be installed in the office, where the records made are under constant observation.

### Electric Iron Smelting.

From time to time notices have appeared in our columns on the Grönwall, Lindblad and Stalhane (A. B. "Elektrometall") process of electric iron smelting, and in our last issue (page 111) we mentioned the starting of a furnace of this type at Trollhattan, Sweden, and gave some of the results obtained with it. It should now be of interest to sum up briefly the progress which has recently been made in the introduction of this process, for which the British, United States, Mexican and other rights have been acquired by Electro-Metals, Ltd., 9½ Union Court, Old Broad Street, London, E. C.

At the meeting of the Iron and Steel Institute in May, 1909, a paper describing this process was read by Mr. E. J. Ljungberg, chairman of the Stora Kopparbergs Bergslags A. B., Sweden, at whose Domnarfvet iron works the experiments had been carried out. Later an exhaustive report on the experiments by Mr. Lars Yngström, managing director of the company, was published (our Vol. VIII, page 11, January, 1910). This report, which gave a complete description of the furnaces, showed that a technically and commercially successful process of electric iron smelting had been developed. We will in the following briefly survey the progress made in the practical application of the process.

In its home country, Sweden, the importance of the process was quickly recognized, more especially because the conditions

of the country are favorable for electric iron smelting. High-grade iron ore is abundant and the numerous waterfalls of the rivers furnish a source of electric power at low cost. In addition the country lacks coal and the iron industry therefore mainly depends on charcoal for fuel which is constantly growing more scarce and costly. The possibility of replacing about two-thirds of the quantity of fuel by electric power together with the prospect of still further improving the quality which has made Swedish iron famous has therefore made the new process a matter of national importance, and it has been taken up as such.

There exists in Sweden an institution called "Jern-Kontoret" which is an association of practically all iron and steel manufacturers in the country. This association is over 150 years old and plays a very important part in the Swedish iron industry. Its object is to supply technical and financial assistance to the members and generally to promote the iron industry of the country.

After careful consideration of the results of the Domnarfvet experiments the board of directors of "Jern-Kontoret" determined to enter into an agreement with the patentees and make an appropriation of money for a plant to thoroughly investigate the process. The fact that this is the first occasion in its history that the association has embarked on a technical undertaking on its own account proves the importance attached to the process. The Swedish Government also offered to assist in the undertaking by supplying electric energy at a nominal cost from their large power plant at Trollhättan.

This installation has now been in operation since the middle of November. It comprises a furnace for 2,500 hp and is elaborately fitted with the best appliances for exact scientific research. It may be safely predicted that the results of work carried out under such auspices will prove of the greatest value and interest.

It is not to be expected that any official report on this work will be available for a considerable time to come, but we are in a position to state that the results so far are highly satisfactory. Some numerical results were given from absolutely reliable private correspondents on page 111 of our last issue. The experience with the experimental furnace naturally suggested various improvements which have been embodied in the new furnace and have proved of great importance for the smooth and economical working of the process.

Meanwhile private enterprise has not been idle. At the conclusion of the experiments by the inventors the experimental furnace at the Domnarfvet Iron Works was taken over by the owners, the Stora Kopparbergs Bergslags A. B., who continued to work it for their own purposes. They are now constructing a new furnace for 4000 hp which is the first of a number of similar furnaces planned for a total of about 40,000 hp. The works contemplated will have an annual capacity of over 120,000 tons of pig iron and the cost is estimated at upward of \$3,500,000.

Another well known Swedish iron works, the Uddeholm Company, have purchased a license and are about to put down a plant of three furnaces for 3000 hp each. Other Swedish works have also acquired licenses for construction and propose to erect works on similar lines.

The new conditions pertaining to electric iron smelting appear likely to foster an iron industry in some countries which have hitherto not produced iron. An example of this is Norway. The ores of this country are mainly of low grade and no coal is found there. Up to the present time no iron has, therefore, been produced there. Water power is, however, abundant and charcoal can be produced. There has now been formed a company which has acquired the Norwegian patent rights to the "Elektrometall" process and is putting down works at Tyssedal near Hardanger. These works will comprise two furnaces for 3000 hp each. Another company has been formed for putting down works at Bilefoss near Arendal, where about 30,000 hp will be utilized for iron smelting.

Similar conditions bid fair to create an iron industry in more distant countries where the cost of freight would enable a native electric iron industry to compete successfully with imported iron and steel. We are informed that Electro-Metals, Limited, are engaged in a number of important negotiations for electric iron smelting plants in several over-seas countries.

The quality of electrically produced iron has proved decidedly superior to that of ordinary pig iron from a similar grade of ore. It will readily be seen that this must be the case, as the sulphur contents can be very much reduced in the furnace and a very dense and tough material is obtained on account of the absence of gases in the iron.

Considering the developments which have already taken place it is safe to predict that this process is destined to play a very important part in the industrial life of many countries and that we may expect to see iron production established in various parts of the world where cheap electric power rather than an abundance of coal determines the prospects of economical success. Canada and California seem particularly in line for such developments.

#### NOTES

**Society of Chemical Industry.**—At the meeting of the New York Section of the Society of Chemical Industry, on Feb. 24, the following papers were read: On "A New Treatment of Water, Preventing Scale in Boilers Without the Use of Chemicals, by Aid of Aluminium Plates," by Mr. Thomas R. Duggan; and on "Boiler Water Purification," by Mr. William M. Booth.

**A Congress of Technology** will be held in Boston on April 10 and 11 to fittingly mark the fiftieth anniversary of the Massachusetts Institute of Technology. A large number of technology graduates, who have been conspicuously successful in varied lines of engineering, will present papers at the congress discussing the conditions and prospects in specific industries.

**The American Mining Congress** has recently increased its board of directors from nine to thirteen, and has adopted a plan for the organization of State chapters. The next meeting place of the congress probably will be in Chicago. Secretary Callbreath is now in the East in connection with the work of the congress, and will arrange for the 1911 meeting.

**Enameled Sills.**—The Stuart & Peterson Company, of Burlington, N. J., has recently completed the job of re-enameled two mammoth chemical stills of Parke, Davis & Company, each weighing 6500 lb. and having a capacity of 650 gal. These stills were originally of German make, but after several years of constant use re-enameled become necessary. To carry it out the Stuart & Peterson Company had to erect a special building and oven. The enamel used is the acid-resisting enamel of the Stuart & Peterson Company.

**Refractory Cement.**—The H. W. Johns-Manville Company, of New York City, have just placed on the market a line of refractory cements for furnace settings of various types, cupolas, lining brass furnaces, assayers' crucibles, oil burning, tilting and rotary furnaces and for patching and facing brick in the fire zone under various conditions. These cements are rated to resist temperatures as high as 3000° Fahr. (about 1650° C.). The same company has also developed a coating for walls known as brickline cement which prevents clinkers from adhering and to seal the pores of the brick.

**Titanium Steel Rails.**—The New York Central Lines have ordered 41,500 tons of titanium bessemer rails for 1911, specifying the use of 1 per cent of the 10 per cent ferrotitanium. This alloy is made, according to the patents of Rossi, by the Titanium Alloy Manufacturing Company, of Niagara Falls. This order will, therefore, require more than 400 tons of the 10-per cent alloy, which is believed to represent the largest single order ever placed for alloy steel rails.



**Instruments of Precision and Laboratory Apparatus** is the title of the handsome new 120-page catalog of Messrs. Wm. Gaertner & Company, of Chicago, giving illustrated descriptions of their numerous instruments of precision (microscopes, telescopes, slide rules, dividing machines) and laboratory apparatus for the demonstration of the phenomena of mechanics, acoustics, heat, light and electricity. Among other instruments Messrs. Wm. Gaertner & Company are the original manufacturers and the co-designers of the apparatus described in Millikan's *Molecular Physics and Heat*, in Millikan & Mills's *Electricity, Sound and Lights*, and in Millikan & Gale's *Laboratory Course in Physics*. The firm has just moved into their new fireproof office and showroom.

**Recording Thermometers.**—We have received from the Bristol Company, of Waterbury, Conn., their illustrated 40-page catalog of Bristol's class III recording thermometers, including a description of the new compensated gas-filled thermometers which are being used for a great variety of applications for recording temperatures up to 800° Fahr.

#### PERSONAL

Mr. Sterling H. Bunnell has resigned his position as works manager of the Grisco-Spencer Company to open an office at 90 West Street, New York, as a specialist in factory system and consulting engineer. Mr. Bunnell is the author of "Cost Accounting for Manufacturing Plants," about to be issued by D. Appleton & Company. Mr. Bunnell has been retained by Messrs. Clinton H. Scovell & Company, certified public accountants of Boston and Chicago, to take charge of the cost keeping and factory efficiency branches of their work and will also represent them in New York.

Mr. J. M. Callow, of the General Engineering Company, Salt Lake City, was in Denver and Leadville, Col., recently on professional business.

Mr. John Collins Clancy has been in Denver several times recently conferring regarding the first installation of his process at Victor, Col.

Mr. J. V. N. Dorr, metallurgical engineer of Denver, has returned home after spending several months in England.

Mr. J. R. Finlay, who has been general manager of the Goldfield Consolidated Mines Company for the past year, has resigned his position.

Mr. H. W. Hardinge, of New York, is on a trip through the West on professional business and for the purpose of investigating the subject of ore reduction, in which he is specially interested.

Mr. Richard Lindenberg and Prof. F. R. Eichhoff are expected to arrive in New York on March 7. Mr. Lindenberg, who has not been in this country since 1904, is the president of the Lindenberg Steel Works, of Remscheid-Hasten, Germany, which were described in detail in our October issue, 1910 (Vol. VIII, page 563), and which are particularly interesting as they were the first German works to introduce on large scale electric steel refining by the Héroult process. Professor Eichhoff was formerly the managing expert of the Héroult-Lindenberg electric furnaces and now occupies the chair of the late Professor Wedding at the Royal School of Mines in Berlin.

Mr. W. H. McFadden has tendered his resignation as vice-president and general manager of Mackintosh, Hemphill & Company, Pittsburgh, to take effect March 1, 1911, and same has been reluctantly accepted by the directors. Mr. McFadden has been closely identified with the rolling mill industry for the past 25 years and has taken an active part in the construction and installation of many of the most important steel mills in this country. Mr. McFadden will divide the next few months between looking after his important gas and oil interests and in traveling for his health.

Mr. William McLaughlin has been appointed mill manager of the Alvarado Mining Company.

Dr. Robert H. Richards recently made a trip into the zinc district of Missouri for the purpose of inspecting the work of the jigs and classifiers designed by him.

Dr. L. D. Ricketts, manager of the Cananea Consolidated Copper Company, Mexico, is taking a vacation in Europe.

Mr. R. W. Robbins has been appointed superintendent of the McKinley-Darragh property at Cobalt, Ontario.

Mr. Dennis Sheedy, vice-president of the International Smelting Company, recently made a trip of inspection to the company's smelter and other properties.

Mr. L. C. Triplett has resigned his position as mill manager of the Alvarado Mining Company, Mexico, and has accepted a similar position at Guanajuato for the Guanajuato Development Company.

#### OBITUARY

Edward Balbach, Jr., president of the Balbach Smelting & Refining Company, Newark, N. J., died on Dec. 30, 1910, after a short illness. Mr. Balbach was born in Carlsruhe, Germany, in 1839, and came to the United States in 1850. Together with his father, Edward Balbach, Sr., he was a pioneer in lead smelting and refining, and in copper refining by electrolysis. He first introduced improvements in lead refining by desilverizing with the addition of zinc, and recovering about 50 per cent of the zinc by distillation. For many years the Balbach works were one of the few sources of refined lead in this country. In copper metallurgy Mr. Balbach did important work in constructing and operating the first electrolytic refinery in the United States in 1881, which was the forerunner of a great industry in which the United States holds the first position. The history of this pioneer American copper refinery was given in our Vol. II, page 303. In the death of Mr. Balbach the metallurgical profession has lost an able and energetic worker who was worthy the high position accorded him by his associates.

Walter L. Pierce, who for thirty-two years had been connected with the Lidgerwood Manufacturing Company and for twenty-nine years its secretary and general manager, died of heart failure at his winter home in the Hotel St. Andrews, New York City, Dec. 10, 1910. He was a son of John F. Pierce, and was born at Dorchester, Mass., on June 8, 1855. His parents survive him, and he leaves a widow, Jane Hutchins; an only son, Walter L. S. Pierce; a brother and a sister. He was known to a wide circle of personal and business associates. He was remarkable as an organizer, and so perfect was his work that no detail of the great business which grew up under his hand was neglected during his long absences from his desk while seeking health, and the coherent body which he formed is a monument to the efficiency of his work. Besides his connection with the Lidgerwood Manufacturing Company he was treasurer of the Hayward Company and of the Gorton-Lidgerwood Company. One of Mr. Pierce's most intimate and life-long friends said of Mr. Pierce: "His unselfish life is his eulogy, and words in lending themselves to praise his steadfast soul do most honor themselves. Given to mingle in the everyday affairs of life, he dignified them in that he added to the world's store of men's confidence in each other. Tenacious in application, indomitable in courage, he was master of his calling in life, but not before he was master of himself. Experience and knowledge were not, however, for him to hoard, capital for accumulation to himself, but rather he gave of himself generously, and no worthy plea so weakly pressed that did not meet a quick response." It will not be out of place to acknowledge here the friendly sentiment which Mr. Pierce always maintained toward this journal since the early days of its foundation.

Charles F. Shelby, general metallurgical superintendent of the copper smelter of the Cerro de Pasco Mining Company in Peru, was killed in an automobile accident between Lima and Ancon, Peru, on Jan. 25, 1911.

Mr. Shelby was among the prominent and able metallurgists in the copper industry. He was born at Omaha, Neb., in 1874. His first professional employment was as assistant chemist at the Globe smelter, Denver. He left this plant in 1900, going to Aguascalientes, Mexico, where he was chemist and later assistant superintendent. In 1903 he entered the field of copper metallurgy under the able direction of Dr. L. D. Ricketts, at Globe, Ariz., where he acted as superintendent of the Old Dominion Copper Company's reduction works. In 1906 he accepted a more responsible position at Cananea, where he inaugurated many improvements in copper metallurgy and aided in bringing the works of the Cananea Consolidated Copper Company to a high state of efficiency. Early last year he was selected as superintendent of the Cerro de Pasco Mining Company, and in company with Mr. A. B. W. Hodges as general manager, went to Peru.

Mr. Shelby was a constant contributor of valuable notes and articles to the technical press, following in this respect the liberal policy of his employers in giving the profession the benefit of his ingenuity and ability.

### Digest of Electrochemical U. S. Patents.

Prior to 1903.

*Arranged according to subject matter and in chronological order.*

*Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.*

#### Ore-Treatment.

61,866, Feb. 5, 1867, Julio H. Rae, M. D., of Syracuse, N. Y. Gold and silver. The crushed ore is placed in the bottom of a glass jar upon a platinum anode plate and around a revoluble platinum-wire cage anode. The dissolved metals are reprecipitated upon a sheet of copper in the solution above the ore.

79,744, July 7, 1868, A. L. Fleury, of New York, N. Y.

Precious metals. Silicious ores, which may be mixed with chemicals, are heated and subjected to chemical gases in a shaft furnace. The treated mass is removed into a closed heated vessel, wherein it is subjected to the action of water, steam and electricity.

90,565, May 25, 1869, William John Lynd, of Golden City, Col.

Removes iron, copper and other discoloring matters from clay by passing an electric current through water containing the clay in suspension.

101,009, March 22, 1870, Nathaniel Haskell, of San Francisco, Cal. Assignor to himself and Joseph F. Steen, of same place.

Copper. The ore is placed in heaps on a hillside where the ground is not porous. A trench is dug around the ore heaps leading to a vessel or vessels below. The ore is alternately drenched with water and exposed to the air for several days. The resulting weak solution of copper sulphate is electrolyzed in the vessels, the copper being preferably electroplated upon metal articles.

239,300, March 22, 1881, Andrew Ryder, Oakland, Cal.

Ore is disintegrated by heating it and dumping it into a liquid carrying an electric current. The apparatus illustrated is a tank containing opposed downwardly converging copper-plate electrodes, the heated ore being dumped between them.

246,201, Aug. 23, 1881, Emile Reynier, of Paris, France. Assignor to Simon Philippart, the son, manager of La Force et la Lumière Société Generale D'Electricite, of Brussels, Belgium.

The ores are used as soluble electrodes in a two-fluid voltaic cell. The anode and cathode compartments are separated by a diaphragm of paper, cloth, skin, membrane or felt. One compartment contains an anode of zinc or lead, or fragments of galena, calamine or other rich conducting ores, a lead or zinc

plate conductor being placed on the ores. The anode solution is caustic soda or potash. The other compartment contains a copper solution, e.g., sulphate. The zinc or lead dissolves and is recovered with regeneration of the caustic alkali. The other metals, e.g., silver, remain as residues. Copper is simultaneously deposited in the other compartment. The cell generates an electric current which is externally utilized. The process is applicable to ores containing also nickel, cobalt, mercury and gold.

272,391, Feb. 13, 1883, Antonin Thiollie, of Villeboeuf de Roche la Moliere. Assignor of one-half to Albert Ancel, of Chateau de Bouches D'Uzure, Canton de Craon, Mayenne, France.

The crushed ores are placed around carbon electrodes in a series of parallel vertical receptacles having perforated walls. The electrodes of opposite polarity are placed between the receptacles with their lower edges in gutters to catch deposits. Beneath the receptacles and electrodes is an open space containing steam-heated coils. The receptacles may be of wood, wickerwork, gutta percha, rubber or coated sheet iron. If the ore is very fine they are lined with cloth or felt. If the ore is a poor conductor coke or plumbago is mixed with it, being separated from the residue and re-used. The cells may have a tight cover to retain fumes. When the electrodes in the ore are anodes, the collecting electrodes between the receptacles may be plates of copper. When the electrodes in the ore are cathodes the intermediate electrodes are insoluble, e.g., carbon. For ores of gold, silver and platinum the embedded electrodes are made anodes. For silver ores a solution of sodium sulphate or nitrate, or of hot sodium chloride may be used, or a cyanide solution. For gold or platinum sulphates nitrates, cyanides, sodium chloride or caustic soda or potash are available. Where the ore contains two or more of these ores sodium chloride is preferably used. If the ore contains silver chloride an ammonia, ammonium chloride, hot sodium chloride or cyanide solution is used. For silver-sulphide ores an ammonia or hot sodium chloride solution is used. For gold telluride ores a sodium chloride or cyanide solution, the ore being placed around the anode, or a sodium chloride solution, the ore being placed around the cathode, in which case the gold deposits at the anode as auric acid. Ores containing gold, silver, lead, copper, iron and metalloids are treated with a hot solution of sodium chloride. The gold, silver, copper and lead deposit at the cathode. Iron remains in the vessel as an oxide. Zinc ores are placed around the anode, ammonia, ammonium carbonate, soda and potash solutions, etc., being used. Ores of lead may be treated with a hot solution of sodium or lead chloride, or with a caustic soda solution. Silver may be parted from lead and zinc. For antimony ores a hot potash solution is used; for mercury ores a solution of the sulphate, nitrate or bichloride of mercury. Copper ores are variously treated, refractory ores being roasted, and ammonia, ammonium carbonate or potash solutions used. Nickel ores are roasted and ammonia or ammonium carbonate solutions used. Ore containing carbonate of iron and calcium may be electrolyzed in a solution of sodium or iron chloride, chemically pure iron being deposited and lime remaining as a residue.

### BOOK REVIEW.

**Welding.**—By Richard N. Hart, B.S. 8vo., 181 pages, 93 illustrations. Price \$2.50 net. New York: McGraw-Hill Book Company.

The four principal chapters are devoted to ordinary welding, electric welding, hot-flame welding, and thermit welding. The descriptions are good, illustrations and photographs first class, and the whole constitutes a useful and instructive book. In his preliminary remarks the author is sometimes hazy in his statements, and his English is at places faulty, but the practical part is well written, clear, and direct. It is a good practical book.